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DEEP LEVEL MINES OF THE RAND

G. A. DENNY

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THE DEEP-LEVEL MINES OF THE RAND

THE DEEP-LEVEL MINES
OF THE RAND

AND

THEIR FUTURE DEVELOPMENT

CONSIDERED FROM THE COMMERCIAL POINT OF VIEW

BY

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AND OTHER MINERALS," ETC.

WITH ILLUSTRATIONS

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PREFACE.

THE writer approaches with some diffidence the questions discussed in this volume. In the light of the experience of the past on the Witwatersrand Goldfields, and in view of the present proportions of an industry which had its beginning only some thirteen years ago—and in view, also, of the lessons of potential development and expansion which it teaches—one feels lost in a maze of magnificent promise. That promise, indeed, stretches far beyond the limit of our present horizon, as bounded by experience and analogy, and included therein are many factors of recognizable uncertainty, adding largely to the magnitude and complexity of the problems which the future may be expected to unravel.

To the writer's mind, the future of these Fields is bound up with the exploration and development of the deeper levels of the Main Reef series; and while not losing sight of the very ample room for expansion still available on the Outcrop Mines, where ores of a low grade prevail, he believes that this latter will be a matter of quite secondary importance in comparison with the probable development in the Deep-levels, in districts where the Outcrop Mines have proved the most remunerative. He therefore confines himself in this volume to the consideration of the future of the Deep-level Mines of the Rand.

The object he has in view is not to discuss issues pertaining to Deep-level Mining from a scientific standpoint, but rather to disclose the bearing they have upon the Gold Mining Industry in its commercial aspect. For this reason, technicality has, as far as possible, been avoided, and the various subjects which are treated will be found (it is hoped) dealt with here in a manner which will be clear not only to the engineer, but to the reader whose training has been purely commercial.

The writer may add that he has in preparation an exhaustive work dealing with the Machinery Equipment of Deep-level Mines.

JOHANNESBURG,
February, 1902.

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CORRIGENDA.

Page 111, *line* 13. *For* Less quartz rock discarded, *read* Less waste rock discarded.

Page 163. *As to item* Native wages, *under heading* TOTAL ESTIMATED SAVING IN WORKING COSTS, *see* POSTSCRIPT *on page* 164A.

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THE

DEEP-LEVEL MINES OF THE RAND.

CHAPTER I.

INTRODUCTION.—EXTENT OF THE GOLDFIELDS.

In speaking of the future of the Rand, there arises before one's mental vision a possibility of productive mines at distances from the outcrop, and at depths from the surface, which to-day would appear impracticable. The proved productive character of the ore in the deeper levels, in all those sections on the Witwatersrand along which the outcrop of the reefs has proved payable, has given data for the projection of similar conditions still further into the deeper ground, with a very great degree of certainty that the postulates will be supported by material realization.

The special features of these banket reefs—inasmuch as they form an integral portion of an enormous aqueous deposit which all competent judges believe to be continuous to any reasonable depth—recommend themselves for permanency and equality of condition; and it is argued, with no little reasonableness, that as the same natural phenomena are responsible for the formation of both banket reefs and quartzite, it cannot be thought likely that the particular conditions which existed to form the banket beds were less general in their effect than those which were in evidence in the laying down of the enclosing quartzites.

The incontestible truths that for 42 miles on the line of strike there is practical continuity of the reef; that in all probability the existing line of outcrop forms to the original detrital edge the relationship of an immensely deep level; that the reefs and quartzites form inseparable parts of a whole which, as such, must be regarded as above any suspicion of doubt of continuity; that the present line of outcrop is only accidental on the plane it represents, and might easily, if acted upon by the same forces in another axial direction, have occupied a position corresponding to portions now very remote from the outcrop;—all these premises form strong links in a chain of evidence, favouring the assumption of the permanency of average conditions in the reefs themselves, which cannot be controverted.

The writer, however, although optimistic as to the future, and although fully appreciating the infinite possibilities inherent in the "deep-deep" zones of the reefs, cannot but realize that for the present any consideration of regions beyond the *third row of deep-levels* would be premature, considering the enormous expenditure of time and money which the development of property within that limit would entail. The principal element of difficulty and uncertainty, in dealing with ground so far removed from present experience as properties distant four miles (if not farther) from outcrop, is our ignorance of the effect which the mighty outburst of igneous rock forming the Klipriversburg range has had upon the horizon of the quartzites and reefs of the Witwatersrand, and of the extent to which the reefs themselves have been affected by the ramifying dykes and numerous faults which would accompany such an outburst. We are in ignorance, also, as to the distance from the eruptive margin at which the reefs may be so affected as to introduce conditions inimical to their economic value.

It has been suggested that the probabilities are in favour of finding that the successive intrusions have so acted upon the reef in the very deep ground, that its depth from the surface (in the sections between intrusions) remains more or less constant. That such conditions exist is quite possible; and granting their existence, the writer holds that the tendency would be to form a series of synclinals, the anticlinal crowns being occupied by dyke matter; and further, that the movements engendered would result in such detailed faulting as to make the working of them (granting for a moment its feasibility) economically impossible.

These are matters, however, which are outside the limit of the third row of "deeps," beyond which it is not proposed to venture in these pages, and they must therefore be relegated to the future.

The Witwatersrand goldfields proper may be stated to extend from the farm Randfontein, distant 21 miles from Johannesburg on the western side, to the farm Modderfontein, distant 21 miles from Johannesburg on the eastern side, both farms inclusive, a total length of 42 miles.

The general direction of a line drawn through the outcrop mines would be east and west, this general direction being subject to slight alteration owing to faulting, and variation in the angles of dip, between the limits of the village of Krugersdorp on the west, and the village of Boksburg on the east, and to more violent alteration at the points indicated; there being at about the position of the first limit named a very radical change to southward, in the direction assumed by the reef outcrop, and similarly a northward inclination at about the position of the second limit.

It is remarkable, *en passant*, that these decided variations in direction—which are due to dynamic causes, in some measure reasonably explicable—should exist at relatively similar positions regarding the wings of the goldfields, and also that at a distance of approximately seven miles, in each instance, from the points at which the change of direction may be dated, the reefs cease to be profitably worked.

Along the whole distance named there is an unbroken line of mines, these being clustered together in the richer sections of the reef, and farther apart in those of lower grade.

CHAPTER II.

GEOLOGICAL CONDITIONS.

(1) Dykes.

Of great importance in the study of the probable value of any deep-level ground is a knowledge of the dykes and fault systems which are present in the neighbourhood, these having a direct bearing not only on mining conditions, but also very frequently upon ore value. The quartzites of the Witwatersrand are ramified by dykes in all directions, from the so-called Hospital Hill shales lying to the north of the reefs—which in the writer's opinion are true intrusive rocks—to the enormous Klipriversburg intrusion forming the limit of the reefs in a southward direction. These bodies of igneous rock may occur in any thickness from a mass 500 feet in width—as in the case of the great dyke which causes the uptilting of the eastern end of the Goch Mine, and the practical verticality of the Henry Nourse—to the veriest wafer thickness, as exhibited in the interbedded dyke between the Main Reef and Main Reef Leader, in the central section of the Rand.

As a rule, the dykes encountered in the mines pierce the quartzite formation in such a manner that they present planes more or less approaching the vertical, although to this rule there are some notable exceptions, especially in the workings of the Violet Mine at Krugersdorp—this mine, however, having only developed the Kimberley reef.

The dykes which, as a rule, cause the greatest difficulty from the aspect of ore-winning, are those which have a course more or less parallel to the strike of the reefs. These dykes frequently cause dislocations of the strata to such an extent that the same reef is twice outcropping at surface, whilst they, in the majority of instances, form faults of some magnitude. The effect of the dykes is well illustrated in the following cross-sections **D** and **E** —Figs. 1 and 2—the first taken through the reef in the vicinity of the East Rand Proprietary Mines, and the second through the New Goch Mine, situated in the central section of the Rand.

The dykes represented in these sections are approximately, in

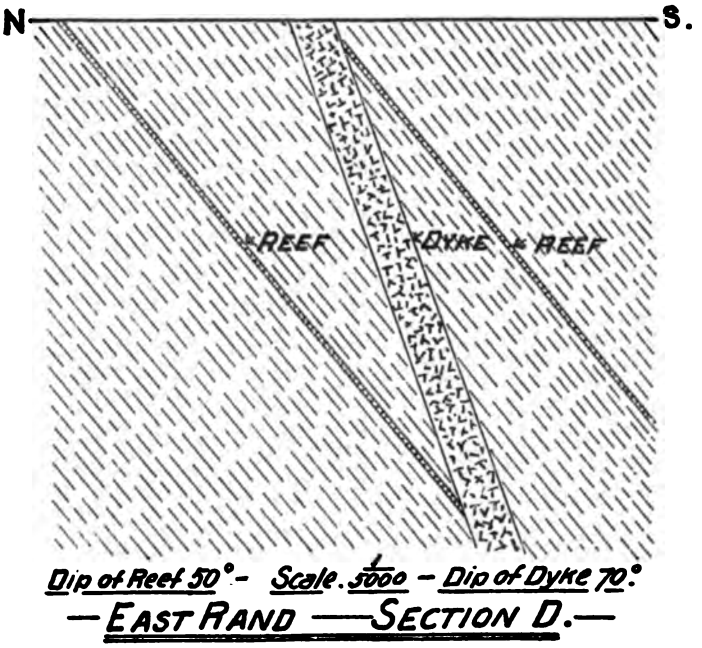


Fig. 1.

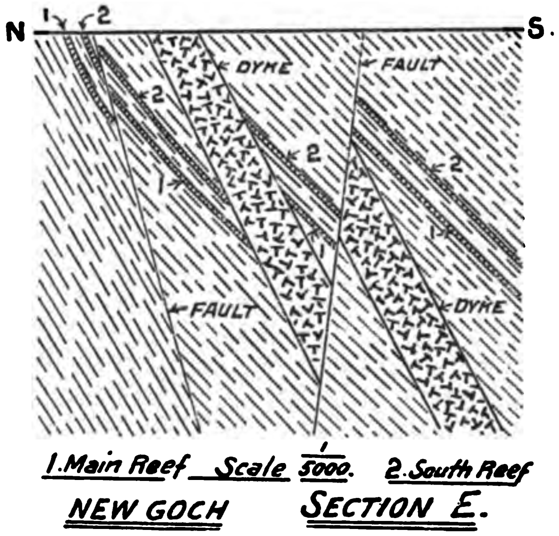


Fig. 2.

horizontal section, 100 feet in width, and as the rock of which they are composed is peculiarly tough to bore, and difficult to blast, cross-cutting through them costs considerable sums of money. It can well be imagined, even by non-technical readers, how such a disturbance of the reefs, as is portrayed in section **E**, interferes with the scheme of mining. The first work would naturally be done on the northern outcropping reef, and the main shaft sunk until the reef was found to be cut off by the dyke, as shown. Experience and applied knowledge would then determine in what position the reef had been thrown by the dyke, and in this case a cross-cut would be run southward for the purpose of intersecting it in its new position.

Later development would prove that the reef was again cut off, and the original cross-cut would probably be continued through the second dyke mass, to again locate the reef's position. The position then could be—and actually is, in the mine which affects this cross-section—that the portion north of the dyke would need separate and distinct development from the portion lying in between the dykes; this portion again requiring the same special development as distinct from the north and south portions, and the southward portion requiring similar separate treatment. In the mine indicated, the writer is sinking a new shaft south of the dyke for the development of the deeper ground. All this separate treatment is, in effect, so much piling up of development costs, which, appearing later in the working costs, reduces the net profit per ton.

The other dykes with which the miner has to deal are those which have a course more or less at right angles to the general reef strike. These, though frequently 100 feet and more in width, do not complicate the mining operations to the same extent as the strike dykes earlier mentioned, and only in rare instances does the writer know of any serious dislocation of the reef on either side of them.

The next aspect of the question of dykes is that of their effect upon values. The writer concedes the view that detail faulting due to dyke action will seriously affect reef grade, for the reason that the earth movements engender frictional sliding of the heavy homogeneous rock masses, which will take effect upon the more porous bodies constituting the reefs, and thus probably in most cases be detrimental to their value. It is not easy to conceive of any possible connection between dykes and the value of the reefs other than in the way previously indicated, if the view be held that the dykes were erupted into their present positions long posterior to the consolidation of the banket reefs, and that in these latter the various constituents of gold, quartz, pebbles, etc., were contemporaneously deposited; and yet it is a singular coincidence that a dip dyke frequently marks the boundary between payable and unpayable ore.

(2) Faults.

Next in importance to the dyke systems, as affecting the mining economy, come the fault systems, and although we have already considered these in combination with the dykes, yet there are features of fault which are quite separate and distinct from dykes. Many faults are, in fact, of much older date, if not really in great measure the primary cause of dykes, for if a portion of the consolidated outer rock shell suddenly detaches itself and presses with enormous force on the inner fluid nucleus, the result would be that the magma forming the said fluid nucleus would burst through the outer shell along planes of weakness to constitute what we know as dykes to-day.

The fault *par excellence* of the Rand is one situated in the Roodepoort district, which is called from its locality the Witpoortje fault. This enormous dislocation has caused a horizontal displacement of the reefs, as between the east and west side of it, of about two miles, and has itself been the subject of very acute differences of opinion as to whether there be any fault at all. Naturally, in some cases, the opinion that the Main Reef series is quite undisturbed in the region of the alleged fault is not quite an unprejudiced one, as the proof of its continuance undisturbed in a westerly direction would mean an exceeding difference in the value of property-holders' areas. But, in the opinion of all scientific observers, there is a unanimity in favour of the existence of the fault and the continuance of the reefs at a point two miles north of the western limits.

Fortunately, there are no other known faults of the same magnitude, though it appears probable that the fault which terminates the reefs on the extreme west, at Randfontein, will prove to be one of unusual proportions.

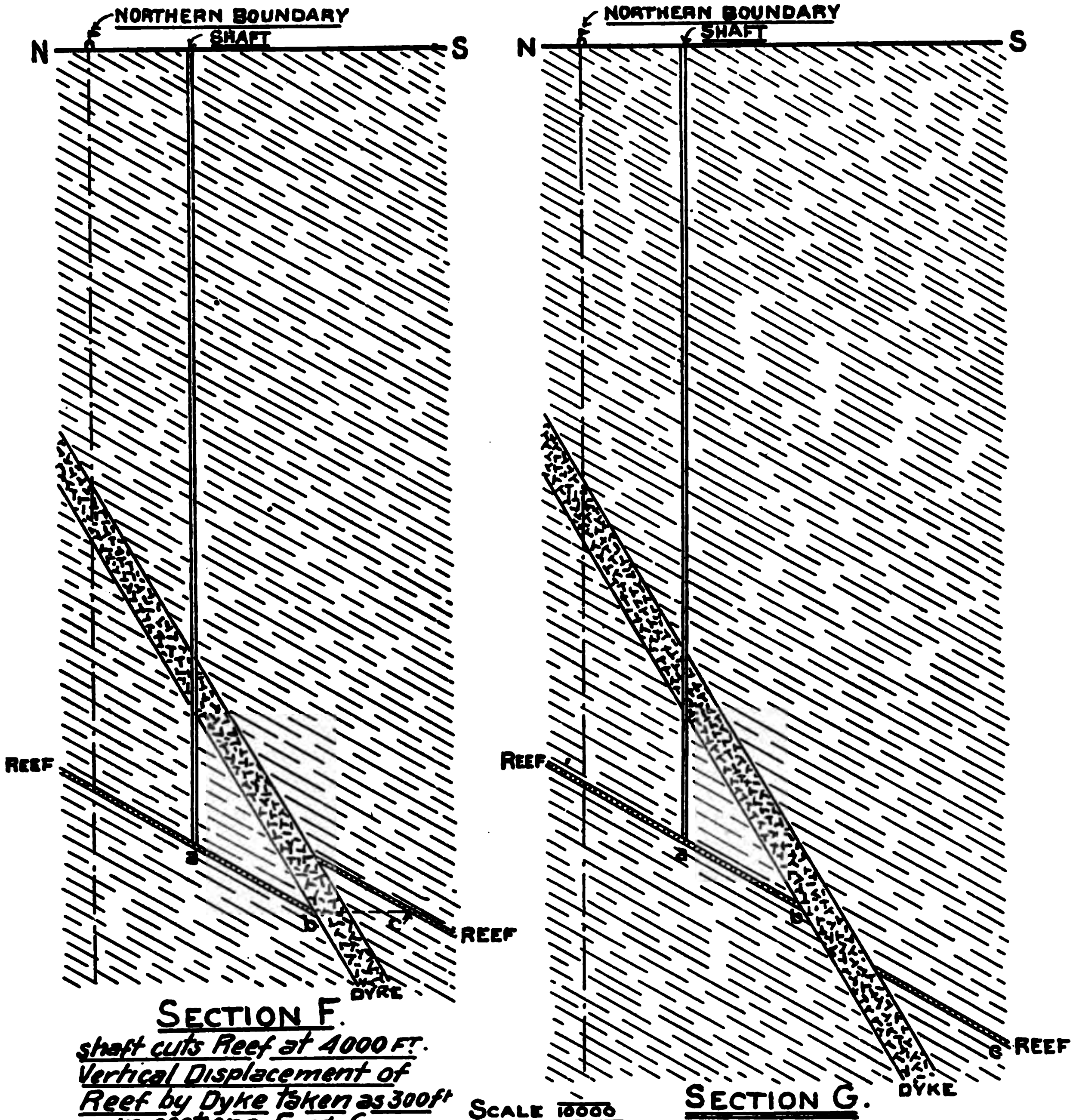
With the exception of the Witpoortje fault, there is no instance within the writer's knowledge of a fault causing a throw of more than 1000 feet, and the majority of even the great faults show less displacement than this. Knowledge of the direction and dip of these great faults, as also of the point of swing, is of immense importance when laying out work either on outcrop or deep-level properties. A fault with a vertical throw of 300 to 400 feet crossing diagonally through a mine is a most serious drawback, for the simple reason that, having the reefs on either side of the fault at an impracticable distance for cross-cutting connections, the two sections have to be considered as separate mines and developed accordingly, necessitating distinct shafts, levels, pumping arrangements, etc., and so greatly increasing the cost.

When a fault occurs which results in a steepening of the dip of the formation, it may happen that, whereas the projection of the fault in the one direction from a given point will show an ever-increasing separation of the reef-planes in depth, its projection in the opposite direction will gradually lessen the separation of the said reef-planes until they finally are first coincident, and then again separated on the opposite wall of the fault. In other words, a fault may be of such a nature that, though causing a heavy downthrow in a given mine, in a neighbouring mine an upthrow may be experienced. It is, for this reason, practically impossible to project faults for any long distance beyond a point of observation, with any certainty of finding the relation of plane to plane constant in respect of the reefs on the one side and the other of the fault.

Another form of fault which is common in the mines of the Rand, is the " strike " or longitudinal fault—that is, a fault which has a direction more or less parallel to that of the reef. This fault may do one of three things: it may cause an inappreciable displacement in either the hanging or footwall directions, it may cause a considerable downthrow, or a considerable upthrow. In the first case its presence is probably quite immaterial; in the second, it will cause considerable loss of ore; in the third, it will sometimes give a duplication of reef, resulting in a substantial gain in the tonnage. In either of the two latter instances, however, it is liable to increase the distance of cross-cutting from the shaft, and therefore the cost of development.

The feature in connection with faulting which has most to be feared in deep-level mining is the occurrence of faults of the nature illustrated in sections **F** and **G** (Figs. 3 and 4, Plate I.).

Here we shall assume that a diamond drill has been utilized to bore from the point *b*, where the reef was lost; and



SECTION F.

shaft cuts Reef at 4000 FT.
Vertical Displacement of
Reef by Dyke taken as 300 ft
in sections F and G.

SCALE $\frac{1}{10000}$

SECTION G.

Plate I. Figs. 3 and 4. [*To face p.* 8.

from the data so obtained, the existence of a downthrow in the one case and an upthrow in the other was determined.

In section **F**, the reef is heavily displaced in the hanging wall direction, and the problem arises, how best to attack it from the vertical shaft. One of three methods would be chosen, namely—

(1) A cross-cut would be run from the shaft at the point *a* to intersect the projected position of the reef at about a point which would give sufficient "backs" for stoping.

(2) A cross-cut would be made from the point *b* to intersect point *c*.

(3) The vertical shaft would be sunk, and the cross-cut between the points *b* and *c* be connected to it.

In the case of section **G**, a new incline shaft would be required, together with all the equipment and attendant expense; new pumping arrangements would also be necessary; and altogether the new section would require separate and distinct treatment as a new mine.

If the method outlined in (1) were followed, the hoisting would remain as partly through the vertical shaft, and partly through the new incline.

If the method outlined in (2) were followed, then the broken ore would need to be handled three times: firstly, through the incline on the side of the dyke remote from the shaft; secondly, through the incline on the nearer side of the dyke; and lastly, through the vertical shaft.

If the method outlined in (3) were followed, the hoisting would remain as in one lift, through incline, from thence through a long cross-cut to the vertical shaft, and to the surface through the latter.

In section **G**, the reef is displaced in the direction of the foot-wall, and one of two methods is open for its exploration, namely—

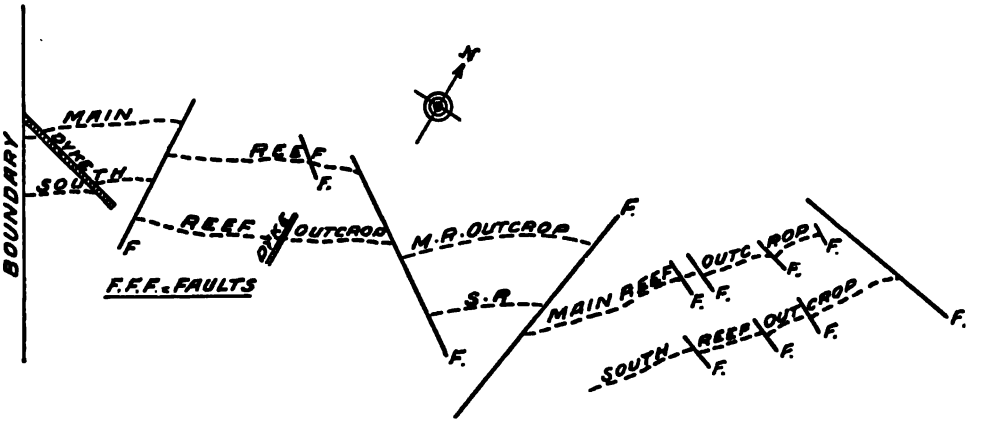
(1) To sink the vertical shaft and cross-cut to the reef.

(2) To steepen the angle of the incline shaft, and thus eventually intersect the reef.

The special method which would be adopted would depend very largely on the judgment of the engineer-in-charge. It is quite clear that in each instance there may arise questions of expediency or exigency which would determine the particular form of attack; in all cases, the very important factor would be that of time.

The delay which heavy faulting occasions in a mine is not only that entailed in the locating of the new position, and the time expended in reaching that position; there is a far more serious delay arising from the loss of development facilities, which are acquired only after gaining a certain number of points of attack. In this case the development of the new sections must be commenced from two to three cross-cuts, a new gallery has to be run, from which the new incline shaft or shafts will begin, the said shaft or shafts must be equipped and sunk, and the whole process of development begun *de novo.* The time and cost which all this extra work means to a company is a very serious consideration.

The annexed illustration (Fig. 5) shows, in plan, a form of



OUTCROPS ON VAN RYN GOLD MINES ESTATE

Fig. 5.

fault that is exemplified in some of the outcrop mines of the Rand, the diagram illustrating the outcrop of the main reef in the property of the Van Ryn Company. The existence of these faults is discovered by the displacement of the line of reef outcrop, and provision can be made accordingly. In the deep-level sections, however, there is not the same facility for observation, and, therefore, it is possible that a company may start operations on a block of claims, through which a diagonal fault passes causing a vertical displacement of 1000 feet.

In the next illustration, Fig. 6, let the property be represented by the lines EW, ER, RS, SW, and let there be a diagonal vertical fault passing through the points E and S. Now assume that shafts are sunk in the positions shown, which strike the reef at 3000 feet vertically, and development commences. The levels driven in the direction of the western boundary would be successively shorter, until the arrival of a time when no more ore could be opened either south or east of the western shaft; in other words, the usefulness of the western shaft as a direct means of ore development on the north side of the fault would have ceased.

The angle of the dip of the fault line, however, as illustrated in Fig. 6, might be such that the western shaft would pass through it before reaching the reef horizon, and consequently this shaft would be useless for the development of ore on the north side of the fault, and it would be, therefore, necessary to sink it 1000 feet deeper, that is, to the horizon of the reef on the south side of the fault.

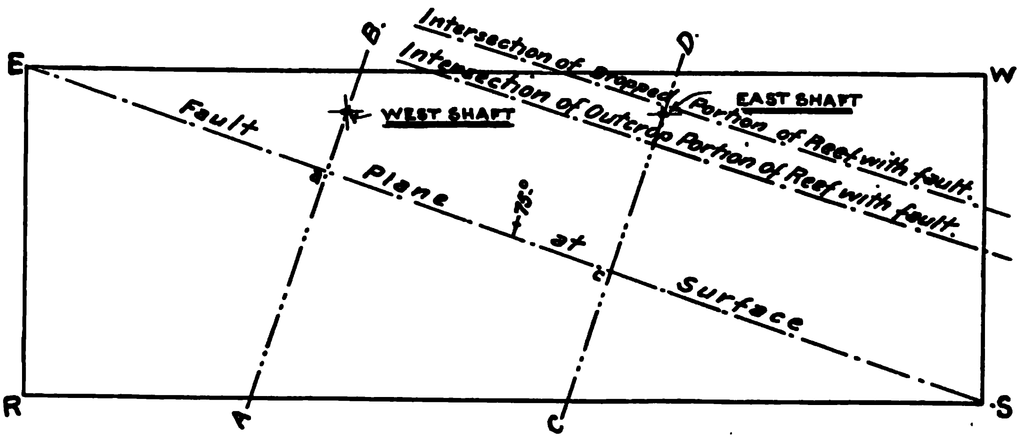
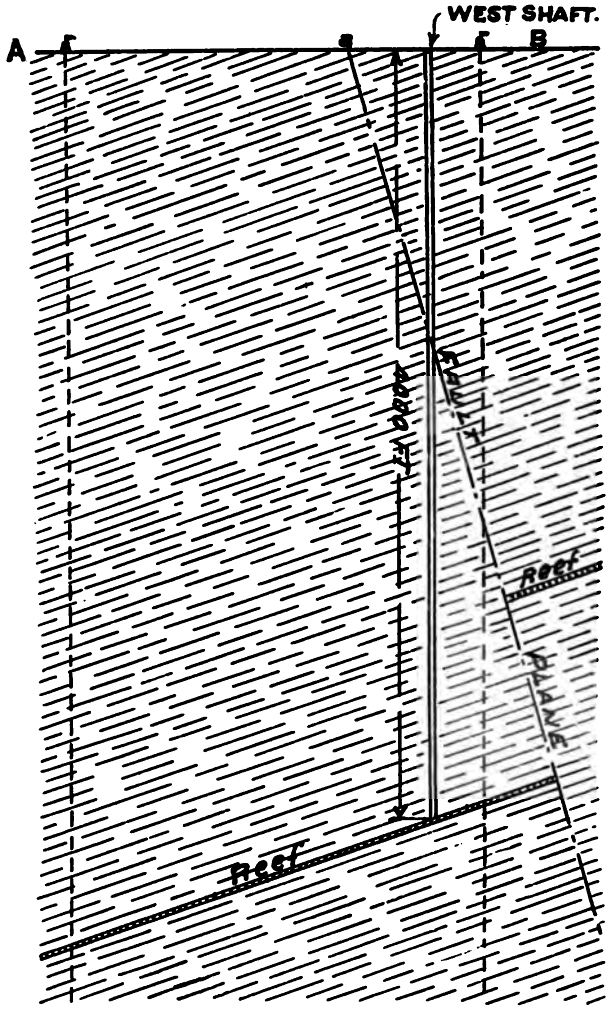

B
D
E
W
Intersection of Dropped Portion of Reef with fault.
Intersection of Outcrop Portion of Reef with fault.
WEST SHAFT
EAST SHAFT
Fault Plane at Surface
75°
a
c
R
A
C
S

Fig. 6.

Section K (Fig. 7) shows the point at which the western shaft would pass through the fault, were it dipping to the north-east at an angle of 75°. The shaft would intersect the fault line at a depth of 1460 feet from the surface, and thereafter would be traversing the country rock on the downthrown or southward side of the fault. The intersection of the fault and reef would be at a point outside of the boundary of the property we are considering, and at that point, the reef would be displaced downwards to the extent of 1000 feet, as shown.

The eastern shaft would strike the reef—which we assume to be dipping at 20°—at 3000 feet vertically, and, as shown in

Fig. 6, would have a small triangular piece of reef to attack on the northern side of the fault. It would, however, be continued to the horizon of the southern portion of reef, and would, as shown in section **L** (Fig. 8), strike the latter again 1000 feet lower, at the intersection of the reef and fault, from which point it would serve to assist in the development of the deeper ground.



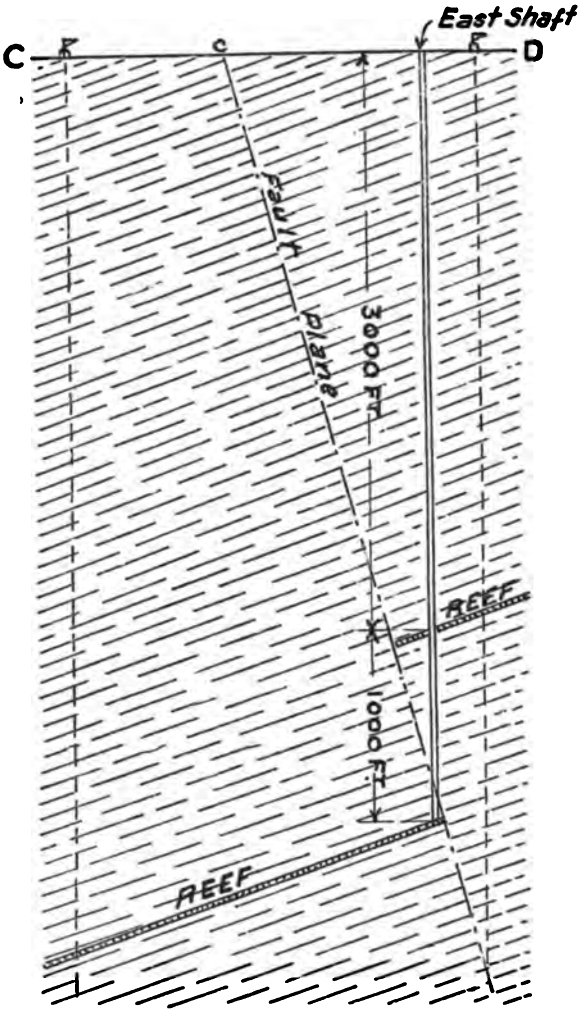
SECTION K.
(ON LINE A.B)

Fig. 7.

The horizontal projection of the fault, in the line of true dip, at a depth of 3000 feet from the surface, is 800 feet, resulting in a duplication of a strip of ore in the region of the eastern shaft

3200 feet in length, by 270 feet in breadth, equal to say 200,000 tons of ore.

The practical effect of a fault dipping, as shown in the foregoing sections, is that the company is confronted with a largely increased expenditure, as well as great delay in reaching the producing stage.



SECTION L.
(ON LINE .C.D)

Fig. 8.

Such cases as that just described illustrate the great necessity that exists, apart from considerations of grade average, for having the development work well in advance of the milling requirement. The delay that would arise in the instances just quoted might easily lead to a suspension of gold production for a period of twelve months, if the precaution of having advance development were not observed; and seeing that during that period there would be no revenue from the mine, the unlooked-for capital sum required would fall very heavily on the company. Moreover, the average cost per foot of the new development would be practically doubled, and therefore would have to be redeemed at a higher figure per ton than ordinarily, resulting in a serious reduction of profits.

CHAPTER III.

CHARACTERISTICS OF THE REEF.

Features of the Reef.

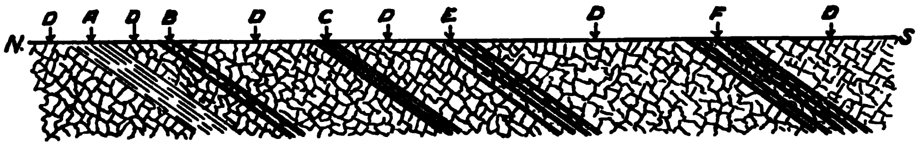
THE singularity of the banket beds of the Transvaal, and the output of gold which they support, have given birth to such a mass of literature on the subject, that only the salient features will be mentioned here. The reefs consist substantially of an arenaceous pyritic base, in which are set rounded pebbles of quartz, quartzite, and, rarely, other rock fragments, consolidated into a massive whole. The quartz pebbles are, for the most part, white in colour, and of the size of pullets' eggs, but they vary both in colour and size within very wide limits. As a general statement, it may be said that the pebbles are quite barren of gold, though cases of its inclusion have been reported from time to time. The gold affects almost exclusively the iron pyrites, which, with the quartzite grains, form the base or cementing medium in the ore, and it may be taken as an axiom, that an ore which contains no pyrites will likewise contain only traces of gold.

On the other hand, however, it does not at all follow that every pyritic ore contains gold; it usually does, but in any proportion, from only a chemical "trace" to the richest impregnation, and hence the presence of pyrites in the ore is, *per se*, no criterion of value. The writer has, however, observed in this connection, that a highly developed crystallization in the pyritic grains—one in which the angles of the crystals are sharply defined—is, as a rule, an unfavourable indication, whilst an imperfect crystallization, in which appear a proportion of rounded forms of pyritic grains, may be regarded as very favourable. The large pellet-like forms of pyrites found in the reef are seen, after oxidation, to exhibit structural rectangular skeleton forms, attached to which may frequently be seen particles of gold.

From the metallurgist's standpoint, the banket ore is singularly free from refractory substances—such as antimony, arsenic, zinc, and lead; though all of these, with the exception of antimony, have been recognized in specimens by the writer, and doubtless all exist in certain small proportions in every mine on these Fields, but the proportion is too small to affect the easy treatment of the ore. Besides the minerals named, platinum and allied metals have been recognized in concentrates obtained from the banket, but again in such proportions as to render their association with the gold only one of scientific interest.

"Visible" gold, or gold which can be seen by the unassisted eye, may always be found in the rich sections of the reef, if trouble be taken to search for it, but, in general, the ore from the mines shows no visible gold.

The gold "reefs" of the Witwatersrand (although, strictly speaking, they do not belong to the same class of deposit as the Australian reefs, and hence the term "reef" is a misnomer, these being stratified beds) are usually recognized as five in number, and in the relative positions shown in the annexed cross-section **A** (Fig. 9).



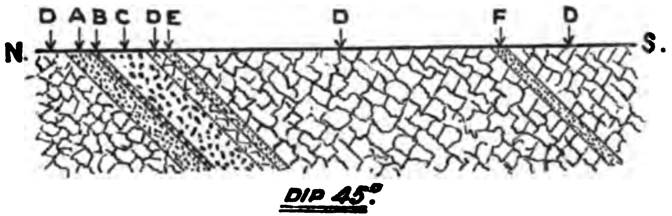
AVERAGE SECTION AT JOHANNESBURG AVERAGE DIP 35°
—SECTION. A.—
A. Hospital Hill Shales (Igneous origin). B. Main Reef Series C Livingstone Reef Series
D Quartzite E Bird Reef Series F. Kimberly Reef Series

Fig. 9.

Distance, horizontal, Main Reef to South reef, from 10 feet to 600 feet.
" " South Reef to Livingstone Reef, from 1500 feet to 2000 feet.
" " Livingstone Reef to Bird Reef, from 1500 feet to 2000 feet.
" " Bird Reef to Kimberley Reef, from 2500 feet to 3500 feet.

The cross-section measured horizontally will vary obviously in strict proportion to the angle of dip which the reefs assume on the line of section, as well as owing to greatly varying widths of quartzite between the reefs. Cross-sections, for instance, taken through the east end of the New Goch Mine, and the western end of the Roodepoort United Mine, show remarkably different thicknesses of quartzite as seen in the sections **B** and **C** (Figs. 10 and 11).

A peculiar feature of the Goch cross-section is the interbedded sheet of igneous rock lying between the Main Reef and Main Reef Leader. This sheet the writer believes to be continuous to the same horizon, from the Robinson Mine on the west to the Van Ryn and Modderfontein Mines on the east, forming in the latter instances the so-called "slate" foot-wall, which is so conspicuous a feature of the mine geology.



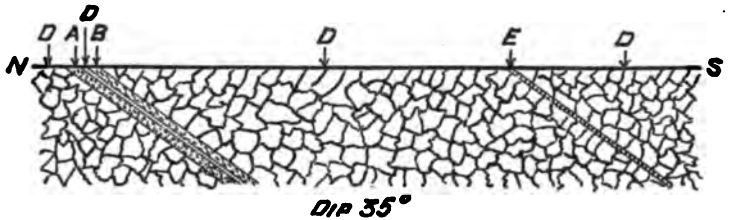
SECTION THROUGH UPPER LEVELS .. NEW GOCH.

— SECTION B. —

A Main Reef. B. Interbedded Dyke.
C. Intermediate Bastard Reef. D. Quartzite
E. Main Reef Leader. F. South Reef.

Fig. 10.

Of the various reefs shown in section **A** (Fig. 9), only the Main Reef, Main Reef Leaders, and South Reef have proved to be consistent in gold value—that is, needless to say, in value sufficiently great to warrant the heavy expenditure required for development and equipment. The Kimberley reef has certainly, at points, been proved to contain payable shoots of ore, notably in the Krugersdorp district, where it is known as the Battery reef. It may be a factor to be reckoned with in the future, on account of its great width and undoubted auriferous character,



SECTION THROUGH UPPER LEVELS. ROODEPOORT UNITED.

— SECTION C. —

A. Main Reef B. Main Reef Leader.
D. Quartzite E. South Reef.

Fig. 11.

but as far as present economic conditions may be taken as a gauge of those that will obtain in the future, we may dismiss it as unlikely to be worked to any extent by the Deep-level companies. In the following pages, therefore, the writer will confine his remarks to the Main Reef, Main Reef Leader, and South Reef, all included in the one generic term " Main Reef Series."

Thickness of the Reefs.

It may well be imagined that, in the stretch of country traversed by the Witwatersrand mines, all variations of width between maximum and minimum are encountered. As a general proposition it may be stated that the Main Reef is the widest body in the series to which it gives its name. It may vary in a given mine, and actually does, in one under the writer's control, from the merest stringer, difficult of recognition, to a massive body of 12 to 15 feet in width. Its average width may be placed at from 4 to 6 feet.

The Main Reef Leader shows similar variation in its thickness as the Main Reef, though at its maximum it never attains the great thickness of the latter. It rarely attains a width greater than 6 feet, and, on the other hand, may not average in a given mine more than 6 inches. Its mean width is probably approximately expressed by 12 to 18 inches.

The South Reef exhibits, in common with the Main Reef and Main Reef Leader, a high degree of lenticularity. It perhaps, more frequently than the other reefs of the series, will change in a development heading, from a solid body of 5 to 6 feet in width to a few disconnected stringers, which are difficult to recognize as the continuation of a fine body, less, in cases, than 100 feet from the point of observation. It undergoes greater changes of character and value, as between district and district, than either of the reefs earlier described. In the central section of the Rand, that is from the Henry Nourse Mine on the east side to the Crown Reef Mine on the west, its width is consistently greater than at any other section of the Rand. It assumes in places within this central district, as in the Meyer and Charlton Mine, a width of 10 to 12 feet, and probably averages not less than four feet. In that section extending westward from the Crown Reef Mine to the Witpoortje fault at Roodepoort, the reef thins out to a mere leader, averaging no more than a few inches in width, and occurring in long stretches as a small stringer not more than one inch in width. From the Witpoortje fault westward it continues as a small body of an average thickness not greater than four inches.

Eastward of the Henry Nourse, and as far as the Modderfontein Mine, the reef is but little developed, on account of poverty. Where exposed through this section, it shows a tendency to divide into a series of thin stringers, this feature being well exemplified in the Van Ryn and Modderfontein properties. Sufficient data are not available to warrant the statement of an average width of reef for this section.

Value of the Reefs.

The values of the reefs, no less than their widths, show every degree of fluctuation between the highest and lowest values recorded. A comparison of the reef values in the different districts of the Rand will disclose the varying tenor of the gold content very clearly. Take, for instance, the comparative values of the South Reef in the Ferreira Mine, and in the May Consolidated. In the first instance, the high-level mark of Rand ore value is reached, in the second it is quite unpayable. The same comparisons may be drawn between either of the other two reefs of this series, with similar results.

In the matter of reef values, the central section of the Rand —or that named as extending from the Crown Reef Mine on the west to the Henry Nourse Mine on the east—can show a better average over the three reefs than at any other section on these Fields. It is not uncommon to find within the limits named that all three reefs of the series—Main Reef, Main Reef Leader, and South Reef—are being mined.

Going eastward from the Henry Nourse, the South Reef and the Main Reef gradually lose their value, until in that section from the Geldenhuis Estate to the eastern end of the fields—Van Ryn and Modderfontein—what is in the writer's opinion the Main Reef Leader only can be profitably mined.

On the western section, going west from the Crown Reef and as far as Roodepoort, the South Reef is the mainstay of the producing mines. Occasional patches of both Main Reef and Main Reef Leader have been worked, but the total percentage of both reefs to the grand total mined is probably under five per cent. From Roodepoort westward, the Main Reef becomes increasingly important, furnishing under its local title of Botha Reef a large percentage of the ore milled in the Krugersdorp district; the South Reef making up the balance. The latest developments in the Randfontein mines lead to the conclusion that throughout that section both Main Reef and South Reef are payable.

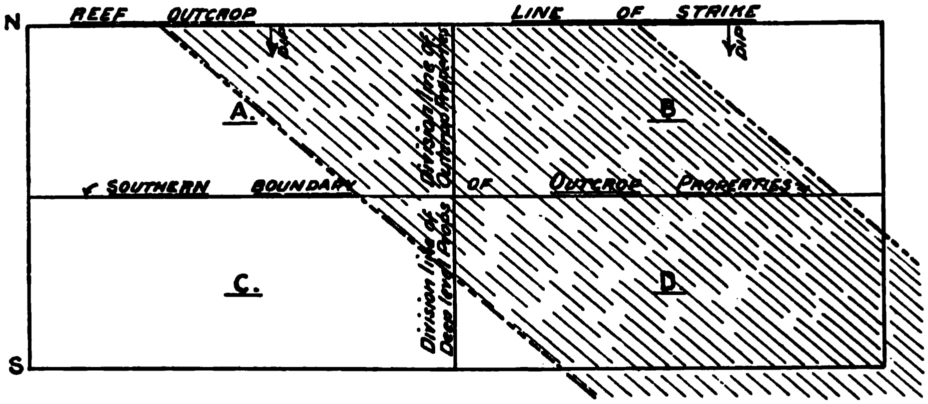
In considering the value of Rand ore, the very common but erroneous idea, that in any given mine but little fluctuation of gold content is experienced, must be abandoned. In this respect, all the features of ore shoots met with in any other class of mineral or metalliferous deposit can be observed, but there are on the whole a greater number of shoots; in other words, there is less unpayable ore, as between ore shoot and ore shoot, in the banket ore of the Rand, than is the case generally with the other deposits mentioned. A glance at the map of the Witwatersrand goldfields will suffice to demonstrate the fact, that there are long stretches of the Main Reef series, which have so far not been very vigorously exploited; and the reason is clearly to be attributed to the low-grade character of the ore within such stretches, since at all points where the reefs are undoubtedly of good grade they are fully worked, to the depths so far attained.

The writer's experience of the features of the ore value in the different mines with which he is actively connected points undeniably to the conclusion that the gold affects certain zones of the banket areas, which said zones have a well-defined direction with regard to the strike line of the reefs, varying in the said direction in the different localities. This feature is of the utmost importance in the writer's opinion, in the determination of the probable values of deep-level property, because the inclusion or otherwise of a rich shoot from an outcrop property might very easily make all the difference between high payability and a struggle to meet expenditure—this being specially the case in the event of having a wide shoot, dipping in a direction widely divergent from the line of dip. The sketch given in Fig. 12 will illustrate what is meant.

Let the figure represent four mining properties, and A and B two companies working about equally productive mines on a wide shoot of ore. In course of time companies are formed to develop the properties of C and D, and the assumption is—

based on the generally accepted view that a property to the immediate dip of a known productive one will be equally productive—that C and D will be equally as valuable as A and B. But later development would prove the value of C to be considerably lower than that of D, because of the direction assumed by the ore shoot.

To explain the *rationale* of shoot phenomena in banket reefs is most difficult, holding the view of the writer that the gold, quartz pebbles, and other constituents of the now consolidated ore, were all contemporaneously deposited. Nature has in some way set up segregating influences in the mass of deposited detritus, which have acted upon the gold particles in such a manner as to induce a more or less constant axial direction in the segregated lines. Under conditions of extreme temperature and pressure—when probably these beds, being less homogeneous than the surrounding quartzites, would become affected, and the silicious matter partly gelatinized—there would be scope for the performance of many natural operations, the nature of which can only be conjectured to-day.


N
S
REEF OUTCROP
LINE OF STRIKE
DIP
DIP
A.
B
C.
D.
SOUTHERN BOUNDARY OF OUTCROP PROPERTIES
Division line of Outcrop Properties
Division line of Deep level Props

SKETCH PLAN SHOWING DIAGONAL TREND OF PAY-ORE SHOOT.

Fig. 12.

If the surfaces of the quartz pebbles have not been gelatinized subsequent to their deposition, how can we reasonably account for the imbedding of pyritic pellets within the surface of quartz pebbles in the manner so frequently seen in specimens from the Kimberley reef? And how can the presence of these pellets—which are not sufficiently hard to withstand the erosion due to transportation from their original source—be accounted for other than by postulating natural segregation? To the writer's mind it is at least clear that, be the scientific explanation what it may, his experience in the development of banket areas points, in almost every instance, to the separation of any given developed area into unpayable and payable zones, these being roughly parallel, constant in a general direction, and invariable as to their classification. As to whether it be permissible to project this theory indefinitely is one on which opinion might reasonably be divided.

In speaking of the value of the Main Reef Series, it must be borne in mind that *values over a stipulated stoping width* are always given. The actual value of the Main Reef Leader and the South Reef footwall leader on a length of, say, 600 feet of drive, may be found to average 10 oz. per ton on clean reef over the total length; but the reef will probably, in such a case, not average more than 3 inches in width, and it may not be possible, owing to the bedding conditions, to take less than 42 inches of thickness when stoping. Thus, instead of having ore assaying 10 oz. per ton to figure on—when calculating the probable recovery value to be obtained from stopes on such a reef—the inclusion of 39 inches of barren rock, necessitated in winning the 3 inches of rich ore, reduces the average value of the whole from 10 ozs. per ton to 14·3 dwts. per ton.

The sorting processes now in vogue again alter the value in proportion to the percentage of waste rock which is discarded; and as this sometimes reaches as high as 40 per cent. of the total ore brought to surface, it is obvious that the assay value of the remainder—which, with the exception of the "fines," is presumably all clean ore—must be considerably increased.

These various influences on the value of the ore proper make it impossible to arrive at original values from a study of the ultimate recovery values which the mines declare. Let us suppose a case of two companies working a thin reef yielding ore of equal value, and assume that "A" company is sorting out 40 per cent. of waste, while "B" company does not sort at all. Let the value of the ore as delivered at the surface be 10 dwts. per ton, then the probable yield from each would be as under:—

"A" Company (40 per cent. sorting) yield per ton milled, 45*s*.
"B" „ (no sorting) „ „ „ 32*s*.

In summing up the value of the reefs of the Rand, it is therefore necessary to know approximately what conditions are present tending to modify the ultimate result, and as these conditions vary in the different mines according to reef width and bedding features, the ultimate values of any arbitrary section can only be ascertained by study of each particular case.

CHAPTER IV.

EXISTING DEEP-LEVEL COMPANIES AND THEIR RECORDS.

Development of Deep-level Mines.

The tendency has been, during the evolutionary period of the Rand deep-level shaft, to demonstrate the necessity for larger and better proportioned shafts than those sunk at the beginning of the said period. The first shafts sunk consisted of three or four compartments, of which two or three were used as hoistways, and one as a pump and ladder-way. The recent practice has been to sink five-compartment shafts, four compartments for hoisting, and one for pump and ladder-way, the latter being extra big for purposes of ventilation.

In the appended illustrations—Figs. 13 to 18, Plates II., III., IV.—particulars are given of a deep-level shaft estimated to strike the reef at 3000 feet, now being sunk under the writer's supervision on the property of the Cinderella Deep, Limited. The details shown in these diagrams will be appreciated by engineers, although perhaps not required for the main purpose of this volume.

Rate of Sinking.

This obviously varies with the conditions which are met with in the sinking, such as water, variations in rock hardness, labour supply, etc. Rates of monthly sinking have been achieved on the Rand which have no parallel in any mining country, as witness the sinking of a 25 feet by 6 feet shaft (inside timbers) by the Simmer and Jack West, Limited, 203 feet in one calendar month, the depth of shaft at the time being over 2000 feet. Every endeavour is made, by introducing systems of bonus for footage performance above a set minimum, to sink

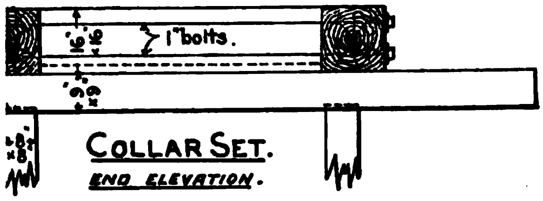
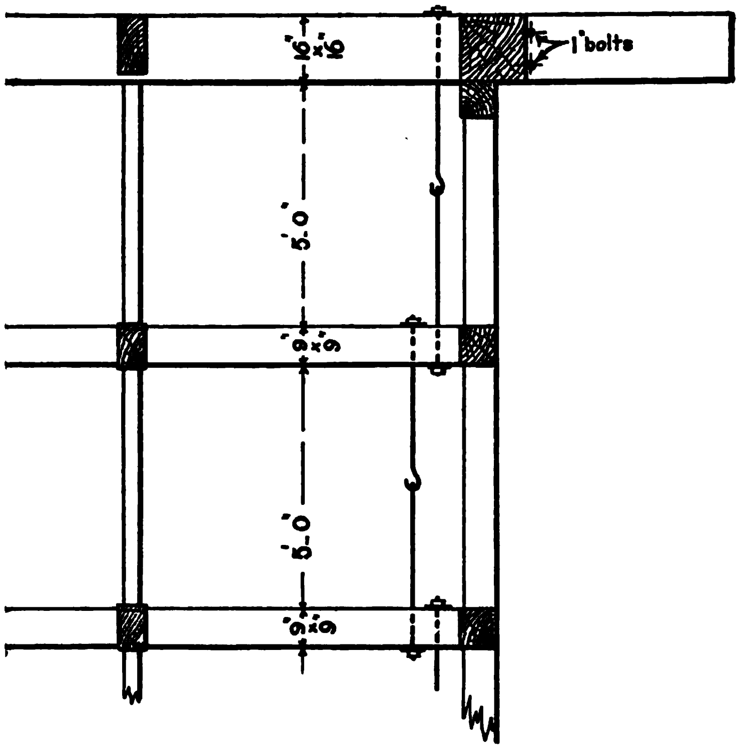
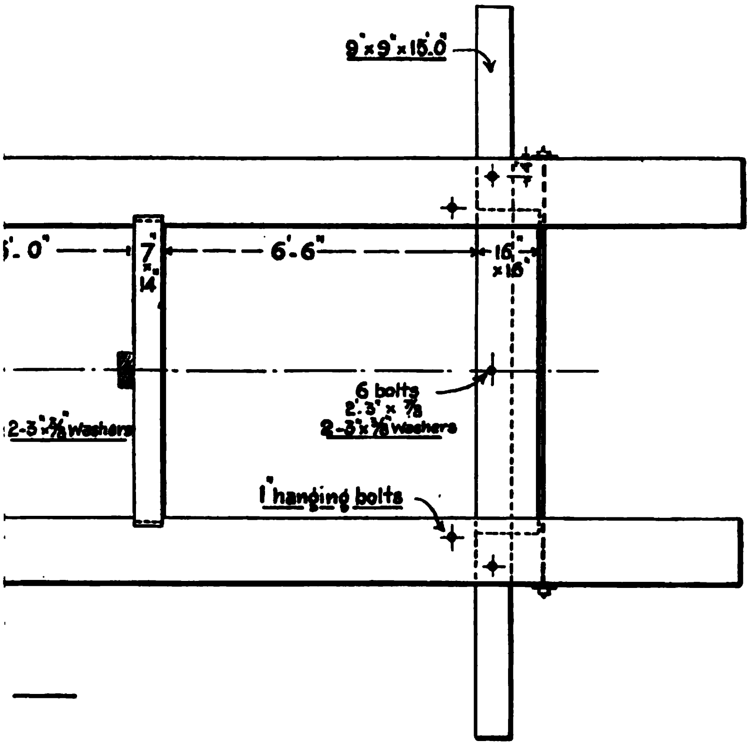


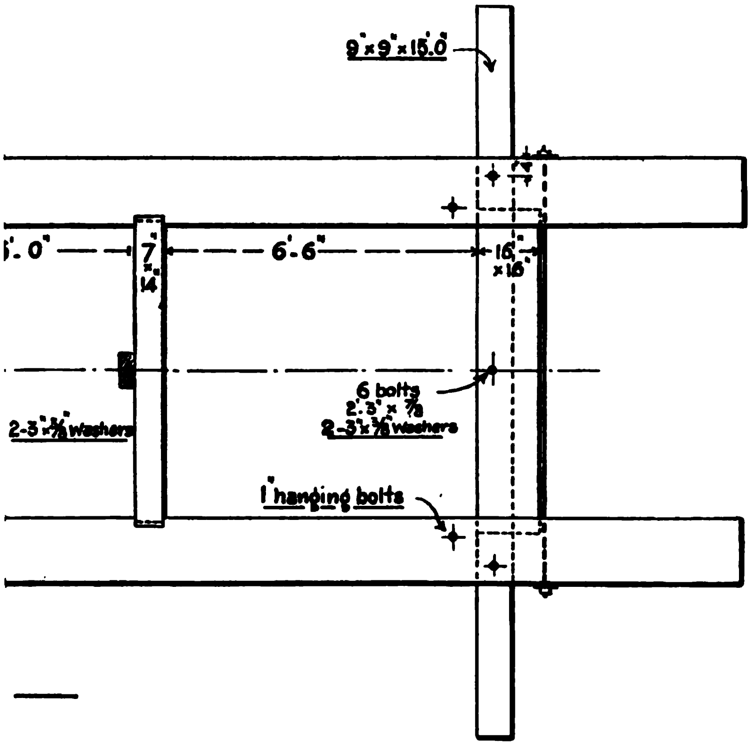
Fig. 14.



Cinderella Deep Limited. [*To face p.* 24.



p Limited (*continued from* **Plate II.**).   [*To face p.* 24.



p Limited (*continued from* Plate II.). [*To face p.* 24.

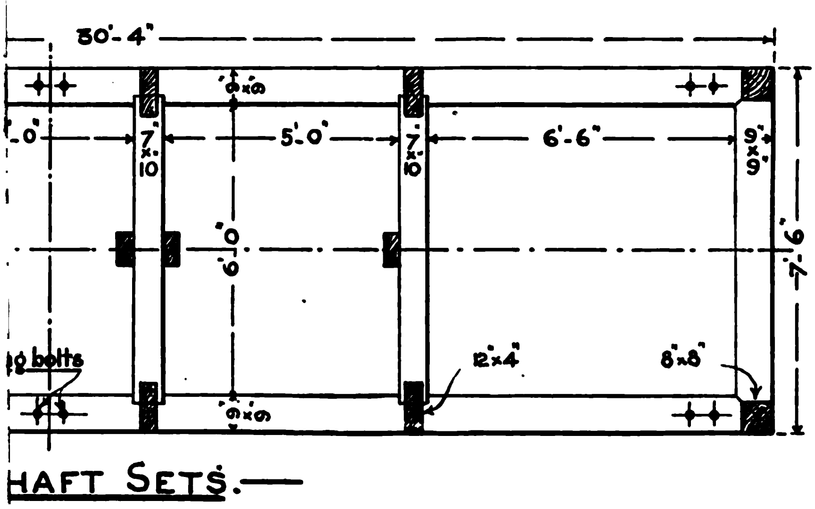


Fig. 17.

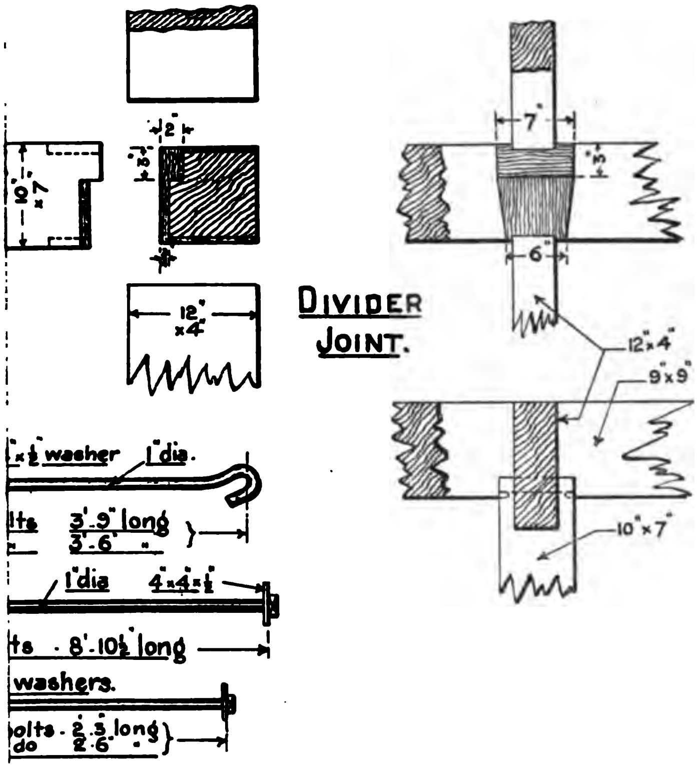


Fig. 18.

at Cinderella Deep Limited (*continued from* Plate III.). [*To face p.* 24.

the shaft at the utmost speed, and in this connection the following table of costs and bonus on several different shafts will prove interesting:—

Ferreira Deep.	Paid as bonus, 6*s.* per foot up to 85 feet, and 12*s.* per foot after that. No bonus given for timbering. Head Office and London expenses are included in Sundry Costs. Native food is included in native wages. Two shafts were sunk during the period.
Angelo Deep.	The shaft is divided into three cage ways, 4′ 6″ × 6′, and a pump department, 6′ × 6′. The timbering is 6″ × 8″ pitchpine, in sets 5 feet apart. A bonus of £1 per foot was paid after 80 feet was made. One shaft only was being sunk.
Simmer West.	The bonus in this shaft was as follows:— From 100 to 120 feet, £6 per foot. From 120 to 140 feet, £7 per foot. From 140 to 160 feet, £8 per foot. All beyond 160 feet, £9 per foot. The sizes of the timbers are as follows:— End and wall plates, 8″ × 8″. Studdles, 3″ × 10″. Dividers, 6″ × 10″. One shaft only was being sunk.
Knights Central.	The managers' and engineers' fees are included in European wages. The bonus was as follows:— From 70 feet to 80 feet, £2 per foot. From 80 feet to 90 feet, £3 per foot. All over 90 feet, £4 per foot. The sizes of the timbers were— Wall and end plates and corner posts, 8″ × 8′. Studdles, 4½″ × 8″. Dividers, 6″ × 8″.

Compiled from official figures by Mr. F. Drake.

It is recognized by the engineers in charge of deep-level works, that supreme importance attaches to the intersecting of the reef in the least possible time, and for that reason every inducement is held out to the workmen for acceleration in the sinking work. The first cost of sinking in these circumstances may appear high, but the value of the increased speed attained, and therefore the earlier starting of development work, easily recompenses a company for the high initial outlay. As an all-round figure, the rate of sinking throughout a deep-level shaft up to 3000 feet in depth may be placed at 115 feet per month. At depths lower than that there will be a diminution in speed, and the writer estimates that from 90 to 100 feet per month will be the highest average rate achieved in the shafts which are laid out to cut the reef at depths of 5000 feet and over. A 6000-feet shaft would therefore occupy, at 90 feet per month, a period of 5·7 years in sinking.

Cost of Shaft Sinking.

This figure varies greatly in the different districts. In the Roodepoort district the cost of sinking is much less than that on the Central Rand, because of inherent differences of structure and bedding in the quartzites. In the same locality, however, very great differences may occur. In one shaft water may be met in more or less abundance, greatly hindering progress; or dykes may be encountered which are very tough to drill, and also require a large expenditure of explosives in blasting; or labour may be scarce, resulting in a low footage sunk, and the consequent higher proportional charges of management and general expenses.

The methods of shaft sinking may be classed under three great heads: (1) Hand work only; (2) Machine drill only; (3) Part hand and part machine work. All three methods have their champions, and doubtless for special conditions a given method may prove more advantageous than others. It has, however, been fairly conclusively proved that in average conditions hand labour is both speedier and cheaper than machine drilling.

Some actual costs on Rand deep-level shafts—compiled by Mr. F. Drake—are shown on the opposite page.

In this class of work, as in all other, gradual reductions in cost were being achieved, and there is no doubt that under better conditions, and with the increasingly marked attention to expenditure in detail which is becoming the rule, the average cost per foot will show a considerable fall.

It may be estimated that the cost of sinking and temporarily equipping a deep-level shaft to about 3000 feet is about £20 per foot, and that the expenditure per foot, calculating in the permanent plant and all work in connection, will not be less than £35 per foot.

On the projected deep-level shafts of to-day—6000 feet and thereabouts—the average cost per foot will be much higher, firstly, because the footage will gradually decrease with depth,

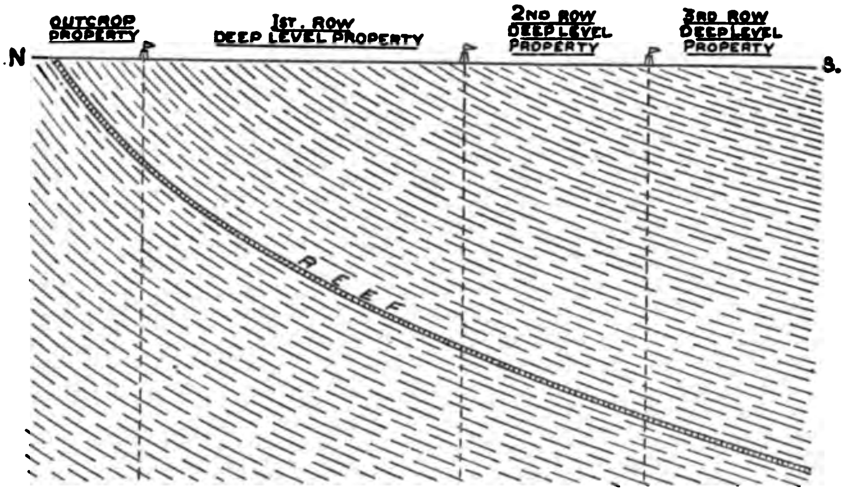
	Ferreira Deep.		**Angelo Deep.**		**Simmer West.**		**Knights Central.**	
Depth of shaft	980–1192		1109–1236 1429–1541		992–1112		1367–1631	
Period covered by return	March and April, 1898		Nov., 1897, and Feb., 1898		March, 1898		Aug. and Sept., 1898	
Depth sunk during period	212 feet		239 feet		120 feet		264 feet	
Size of shaft within timbers	21½ feet × 6 feet		21 feet × 6 feet		25 feet × 6 feet		26 feet × 6 feet	
	Cost per foot, shlgs.	Per cent.	Cost per foot, shlgs.	Per cent.	Cost per foot, shlgs.	Per cent.	Cost per foot, shlgs.	Per cent.
SINKING.								
European wages	37·23	15·16	48·73	14·48	61·62	15·02	57·04	16·53
„ bonus	7·18	2·92	19·63	5·89	27·50	6·71	33·03	9·55
Native wages	82·08	33·44	69·88	20·71	72·25	17·62	90·13	26·09
„ food	—	—	13·56	4·25	28·03	6·83	11·48	3·32
Dynamite	22·87	9·31	28·37	8·42	32·55	7·94	20·18	5·84
Fuse	1·43	0·58	1·87	0·56	1·25	0·30	1·70	0·49
Detonators	0·52	0·21	0·87	0·25	0·53	0·13	0·45	0·13
Coal	14·85	6·23	12·25	3·63	21·85	5·33	26·95	7·80
Lubricants, packing and waste	1·45	0·59	1·61	0·47	0·15	0·04	1·02	0·29
Tools	0·59	0·24	2·26	0·67	1·55	0·38	—	—
Iron, bolts, nuts, etc.	0·32	0·13	1·03	0·31	0·77	0·19	1·85	0·53
Paraffin, candles, etc.	1·52	0·62	3·93	1·16	2·27	0·55	1·45	0·42
Sundry stores	1·55	0·63	4·56	1·35	9·28	2·26	4·00	1·16
Workshops account	1·35	0·55	—	—	—	—	—	—
Drill steel and sharpening	12·08	4·92	—	—	0·85	0·21	—	—
Sundry cash expenditure	1·70	0·69	2·92	0·86	—	—	—	—
General maintenance	—	—	7·33	2·17	36·24	8·84	—	—
TIMBERING.								
European wages	6·46	2·62	16·20	4·81	12·24	2·98	15·99	4·62
„ bonus	—	—	—	—	10·00	2·44	—	—
Native wages	1·24	0·51	—	—	3·66	0·89	2·51	0·72
„ food	—	—	—	—	1·26	0·31	0·31	0·09
Timber	47·18	19·22	—	—	47·17	11·50	53·57	15·50
Sundry stores	0·54	0·22	41·48	12·32	4·25	1·04	1·50	0·43
Workshops account	4·31	1·75	—	—	—	—	—	—
HEAD OFFICE AND LONDON.								
Managers' and engineers' fees	—	—	34·86	10·35	13·33	3·25	—	—
Directors' fees	—	—	13·94	4·14	9·72	3·25	—	—
H. O. salaries	—	—	5·76	1·71	6·66	1·62	6·06	1·75
London salaries	—	—	5·71	1·69	5·14	1·25	3·78	1·00
TOTALS	246·45	100·00	336·45	100·00	410·12	100·00	345·62	100·00

because of the delay that will arise in clearing the bottom of the shaft after each blasting; and, secondly, because all equipment will require to be of a much heavier and more expensive type.

The writer would estimate an increase of £5 per foot on the second stage of the shaft-sinking, that is, from 3000 to 6000 feet, making the cost per foot without the full equipment £25 per foot, and with the full equipment £40 per foot.

Existing Deep-level Mines.

In common Rand parlance, the deep-level mines now producing gold belong to the "first row" of deeps, the said rows being extended indefinitely, according to the fancy of the respective property-holders. The north boundary of the first row of producing deeps averages only a little over 1000 feet from the outcrop; the second, 4000 to 5000; and the third, about 6000 to 7000 feet from the same starting-point. Section **M** (Fig. 19) is a section from the outcrop to the third row of deep levels.


OUTCROP PROPERTY
1ST. ROW DEEP LEVEL PROPERTY
2ND ROW DEEP LEVEL PROPERTY
3RD ROW DEEP LEVEL PROPERTY
N
S.
REEF

SECTION M.

Fig. 19.

The varying angles of dip encountered along the Main Reef outcrop, make it quite impossible to state a general average for the depths of shafts in the first, second, or third row of deeps. This is well illustrated in the case of the Nourse Deep shafts, respectively but 520 and 820 feet from the outcrop. In the first case, the South Reef was struck at approximately 1300 feet deep, and in the second, the same reef was intersected at slightly over 1400 feet. In contrast to this, the deep-level shafts of the West Roodepoort Deep, and the Roodepoort Central Deep, distant from outcrop 2500 and 2600 feet respectively, strike the South Reef in the first instance under 1200 feet, and the second under 1400 feet; and, again, the Vogelstruis Consolidated Deep West shaft, 2190 feet from outcrop, or somewhat less than the Roodepoort shafts, did not strike the South Reef until a depth of 2400 feet was reached. In each instance, therefore, separate data have to be collected, upon which to base estimates of depth.

The first row of producing deep-level mines occupies an aggregate length of 12·5 miles, out of a total length, from Modderfontein on the east, to Randfontein on the west, and neglecting all curves on outcrop, of 42 miles. From the total length, let 10 miles be deducted to allow for poor and badly faulted ground, and there remains a length of say 32 miles. The aggregate of 12·5 miles, representing the producing deep-levels, forms to the total length of reef as just specified, a proportion of 40 per cent. nearly.

At the moment there is no deep-level mine in existence—or projected, as far as the writer is aware—beyond the limits of Roodepoort on the west, and Boksburg on the east, with the exception of the Tudor deep-level to the west of Roodepoort, and the Geduld Proprietary on the east of Boksburg. The Benoni Company is not, strictly speaking, a deep-level, nor can the Modderfontein Company, although possessing a vertical shaft to the south of the Van Ryn Company, be properly classed as a deep-level, as it also works the outcrop of the Van Ryn Reef at the eastern end of its property. As there is, therefore, but little evidence of deep-level values beyond the limits stated, we might fitly, for the present, confine our consideration to them.

The length of a straight line drawn from the West Roodepoort Deep, which is on the western limit, to the Cinderella Deep, which is on the eastern limit, of the section of the Rand we shall proceed to consider, is approximately 25 miles. The total number of deep-levels actually at work, either producing, in the development, or the shaft-sinking stage, is—going from east to west—33:

Name of Company.	No. of claims.	Position.	Present stage.
Cinderella Deep	290	2nd row	Shaft sinking
Angelo Deep	306	2nd ,,	,, ,,
Witwatersrand Deep	273	1st ,,	Developing
Knights Deep	179	1st ,,	,,
Knights Central Deep	443	2nd ,,	,,
Glen Deep	183	1st ,,	Producing
Simmer and Jack East	297	2nd ,,	Developing
Rose Deep	181	1st ,,	Producing
South Rose Deep	265	2nd ,,	Developing
Rand Victoria East	273	3rd ,,	Shaft sinking
South Geldenhuis Deep	192	2nd ,,	,, ,,
Rand Victoria Mines	299	3rd ,,	,, ,,
Geldenhuis Deep	212	1st ,,	Producing
Simmer and Jack West	200	2nd ,,	Shaft sinking
Jumpers Deep	257	1st ,,	Producing
Jupiter G.M. Co.	299	2nd ,,	Shaft sinking
Nourse Deep	202	1st ,,	Producing
South Nourse Deep	290	2nd ,,	Shaft sinking
Village Main Reef	140	1st ,,	Producing
Village Deep	186	2nd ,,	Shaft sinking
Ferreira Deep	141	1st ,,	Producing
Robinson Deep	209	1st ,,	,,
Robinson Central Deep	45	1st ,,	Shaft sinking
Crown Deep	169	1st ,,	Producing
Langlaagte Deep	184	1st ,,	,,
Langlaagte Royal	107	1st ,,	Equipped, but not producing
Main Reef East	243	1st ,,	Shaft sinking
Main Reef Deep	245	1st ,,	,, ,,
Main Reef West	343	1st ,,	,, ,,
Vogelstruis Cons. Deep	376	1st ,,	Developing
Durban Rood. Deep	232	1st ,,	Producing
Roodepoort Central Deep	186	1st ,,	,,
West Roodepoort Deep	244	1st ,,	Developing

This list classified stands as in the next table.

First Row Deep-levels.

No. of Companies.	Producing.	Developing.	Shaft Sinking.
21	12 1 equipped, but not producing	4	4

Second Row Deep-levels.

10	—	3	7

Third Row Deep-levels.

2	—	—	2

In the area under consideration—that is, from the Cinderella

Deep on the east to the West Roodepoort Deep on the west—these figures show that in the position of the first row of deep-levels, of a total of 33 companies in a length of 25 miles, only 13 have so far reached the producing stage; one of which, the Langlaagte Royal, has been shut down since November, 1896, the mine being so broken up by dykes and faults as to make the operations unprofitable. In the position of the second row of deeps, only three companies have reached the developing stage, though this is imminent in the cases of the Jupiter Gold Mining Company, Simmer and Jack West, and South Geldenhuis Deep. In the position of the third row of deeps, only two companies have so far broken ground.

Assuming an average length of 6000 feet along the strike as the basis of a company's holding, there is room in the distance, between the limits taken, for 22 companies. Out of this length however, because of insufficient data, must be deducted all that stretch of reef from the Langlaagte Deep Westward to the Durban Roodepoort Deep.

The length of the section named is approximately eight miles, which, deducted from the total distance between the stated east and west limits, leaves a length of 17 miles; and of the total 33 deep-level companies scheduled on an earlier page, no less than 28 are included in the 17 miles with which it is now proposed to deal.

Deep-level Shafts now Sinking.

The following are the deep-level shafts now in the process of sinking, the approximate distance from the outcrop, and estimated depth to the reef.

SHAFTS LESS THAN 2000 FEET DISTANCE FROM OUTCROP.

Company.	Shaft.	Distance from Outcrop.	Present depth.	Estimated depth to reef.
Main Reef West . . .	East	1900	50	1300

SHAFTS DISTANT 2000 FEET FROM OUTCROP AND UNDER 3000 FEET.

Main Reef Deep . . .	West	2040	70	1250
Vogel. Cons. Deep . .	West	2190	2400	2500
Kleinfontein Central . .	West	2300	370	1160
Robinson Central Deep .	Main	2320	500	1550
Main Reef	East	2435	80	1700
Main Reef East } Main Reef Deep } . .	Joint	2700	40	1800
South Nourse Deep . .	West	2870	80	2300

Shafts distant 3000 feet from Outcrop and under 4000 feet.

Company.	Shaft.	Distance from Outcrop.	Present depth.	Estimated depth to reef.
Kleinfontein Central . .	East	3030	250	1620
South Nourse Deep . .	East	3030	80	2370
Village Deep . . .	East	3100	1200	2250
Tudor	East	3210	only started	2230
Tudor	West	3210	only started	2230
Village Deep . . .	West	3400	700	2250

Shafts distant 4000 feet from Outcrop and under 5000 feet.

Lancaster	Central	4000	100	1900
Cinderella Deep . . .	Main	4000	550	3000
Lancaster West . . .	Botha Deep	4100	100	2000
South Geldenhuis Deep .	Milner .	4160	2750	2800
Angelo Deep . . .	West	4260	2430	4100
Angelo Deep . . .	East	4350	2400	4100
South Geldenhuis Deep .	Rhodes	4500	2000	2800
South Rose Deep . .	Rudd	4540	2400	3100
Knights Central. . .	East	4937	650	2300

Shafts distant 5000 feet from Outcrop and under 6000 feet.

Simmer and Jack East. .	Lohse	5080	1780	1830
Simmer and Jack West .	Howard	5340	3370	no estimate
Simmer and Jack West .	Webb	5640	430	3000
Jupiter	Sapte	5650	270	3000

Shafts distant 5000 feet from Outcrop.

Jupiter	Cablin	6280	3750	no estimate
Rand Victoria . . .	Albert	8090	only started	4000
Rand Victoria . . .	Union	8100	" "	4000
Rand Victoria East . .	Victoria	8150	" "	4000

These shafts are conveniently classified in depths thus:—

Depths from surface, in feet.	1500 and under.	2000 and under.	2500 and under.	3000 and under.	3500 and under.	4000 and under.	4500 and under.	No estimate.
No. of shafts	3	7	8	5	1	3	2	2

Total 31 shafts.

In the length of 17 miles which we have taken as a basis for the operations of deep-level companies, the first row of deep-levels, comprising 21 companies, may be said to be all in process of active exploitation, as in these areas—notably in the East Rand—where first rows of deep-levels do not exist as separate companies, the ground is held and under operation in the upper levels by the outcrop companies. In the second row 10 companies are at work, and in the third only two. Assuming that in the second and third rows the properties will average 6000 feet along the strike of the reef, there remains room for a further 17 companies, making in all, on this section alone, a total of 51 companies.

Records of Deep-level Mines.

For evidence of permanence in the conditions of the reefs, we cannot do better than study the records of the *producing* deep-level mines. These are twelve in number, namely :--

West of Johannesburg	Roodepoort Central Deep. Durban Roodepoort Deep. Langlaagte Deep. Crown Deep.
South of Johannesburg	Robinson Deep. Ferreira Deep. Village Main Reef.
East of Johannesburg	Nourse Deep. Jumpers Deep. Geldenhuis Deep. Rose Deep. Glen Deep.

COMPARATIVE YIELDS, COSTS, AND PROFITS.

OUTCROP COMPANIES AND CORRESPONDING DEEP-LEVELS.

OUTCROP COMPANIES.				CORRESPONDING DEEP-LEVELS.			
Name of Company.	Yield per ton.	Cost per ton.	Profit per ton.	Name of Company.	Yield per ton.	Cost per ton.	Profit per ton.
Princess Estate .	34·92	28·0	6·92	Roodepoort Central Deep . .	31·90	30·9	1·00
Durban Roodepoort .	50·00	27·7	22·30	Durban Roodepoort Deep . . .	48·97	36·4	8·57
Langlaagte Estate .	27·75	20·6	7·15	Langlaagte Deep .	33·56	28·0	5·56
Crown Reef . .	52·47	23·9	28·57	Crown Deep . .	36·95	23·2	13·75
Robinson . . .	71·76	23·0	48·76	Robinson Deep . .	47·59	34·4	13·19
Worcester . .	45·04	21·0	24·04	Ferreira Deep . .	70·00	31·9	38·10
Ferreira . . .	85·22	27·5	57·72				
Wemmer. . .	66·32	24·9	41·42				
Salisbury . . .	27·23	21·0	6·23	Village Main Reef .	65·10	26·9	38·20
Jubilee . . .	34·00	21·0	13·00				
Henry Nourse . .	57·45	26·9	30·55	Nourse Deep . .	44·08	30·5	13·58
New Heriot . .	46·28	25·5	20·78	Jumpers Deep. .	41·73	30·0	11·73
Jumpers . . .	33·96	24·8	9·16				
Geldenhuis Estate * .	46·24	17·8	28·44	Geldenhuis Deep .	42·07	21·0	21·07
New Primrose . .	33·46	18·5	14·96	Rose Deep . .	42·04	21·4	20·64
May Consolidated .	42·43	19·8	22·63	Glen Deep . .	44·42	26·8	17·62
Glencairn . .	29·92	17·6	12·32				

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* The Stanhope and Geldenhuis Main Reef Companies, which, jointly with the Geldenhuis

An examination of this table will show that there is not of necessity the same grade ore in the outcrop and immediate deep-level ground—as witness the Crown Deep and Crown Reef—the tendency being to find a similarity between the outcrop mine and its next eastward deep-level; in other words, the trend of the ore shoots has a direction rather eastward of the true dip line, in all that region from the Langlaagte Deep to the Glen Deep. In the Roodepoort district the shoot direction is westward, although this is not clearly shown by the results so far achieved by the deep-levels; but the writer, in examining mines in that district, and recommending the policy of development, has based the said policy on that view, with quite successful results, in opening up ore of higher grade within the projected shoot direction than in other sections.

The results shown in the table, however, go to show that in the neighbourhood of rich outcrop mines there is also rich ore in the deep-levels. Taking the districts "on grade," the central district comes easily first with the high-grade yields of the Ferreira Deep, Village Main Reef, and Robinson Deep, whilst there is quite a remarkable uniformity of grade in the more easterly mines, from the Nourse Deep to the Glen Deep, both inclusive. The Langlaagte Deep is probably the lowest grade of the series, although it shows better in the list than the Roodepoort Central Deep; but the writer believes, as elsewhere mentioned, that the grade of the latter, as shown, is by no means a fair criterion of the value which the mine will yield when working under normal conditions.

In the comparison as it stands, the *yields* from five of the deep-level mines are lower than those of the corresponding outcrop mines, and seven show a higher return.

In the matter of *cost per ton*, the outcrop mines easily hold the low record, and only in one instance, that of the Crown Deep, is there any approximation to similarity as between the outcrop and the corresponding deep mine.

Taking the differences, in each case, between yield and cost, there are only two instances—namely, that of the Village Main Reef and the Rose Deep—in which the result is appreciably in favour of the deep-level mine. In one other instance, that of

---

Estate, work the outcrop along this section, are not included, as, on account of their small area and the small scale of their operations, they form no basis of comparison.

the Glen Deep, the "difference" is almost practically the same as the average of its outcrop mines.

Averaging the differences between yield and cost per ton of the outcrop mines, and those of the corresponding deep-level mines, and comparing the figures, the result is 6*s.* 3*d.* per ton profit in favour of the outcrop mines, and in like manner, a comparison of the average costs will show that the outcrop mines work for 5*s.* 6*d.* per ton less than the deep-levels. No comparison of either cost or profit is, however, quite fair, because of differences in amount of waste rock sorted.

It must be taken into consideration, however, that at least three of the deep-level mines are not working normally—namely, the Roodepoort Central Deep, the Langlaagte Deep, and the Ferreira Deep.

In the first case, that of the Roodepoort Central Deep, the published assay value of the ore developed in the mine, taken together with the high percentage of sorting, calls for a recovery equal to that of the Durban Roodepoort, namely, 50*s.* per ton.

The reason this was not obtained was due entirely, it is alleged—and the writer believes with good foundation—to shortness of native labour, making it necessary to use machine drills in the stopes, thus increasing the width very considerably above that of hand stoping; and further, in order to augment the mill supply, about 35 per cent. of the ore milled was obtained from the development dumps. These explanations are quite sound when it is considered that the actual average width of the reef in this mine is probably under 6 inches. Given a full complement of native labour, and other conditions favourable, the mine should produce 50*s.* per ton, if any reliance can be placed on the published assay values.

In the second case, that of the Langlaagte Deep, the costs had been reduced, in the last two months of running, by about 4*s.* per ton.

In the third case, that of the Ferreira Deep, the mill had only been running three months, and, without doubt, the costs would have been considerably lowered when the mine and plant had commenced to run smoothly.

On the next page will be found a detailed tabular statement of the returns given by the various deep-level companies for the periods shown.

RETURNS GIVEN BY THE VARIOUS DEEP-LEVEL COMPANIES FOR THE PERIODS SHOWN.

Name of Company.	No. of stamps.	Tons milled.	Total value of yield.			Value of yield per ton.		Total cost.			Cost per ton.		Total profit.			Profit per ton.		Period covered.
			£	s.	d.	s.	d.	£	s.	d.	s.	d.	£	s.	d.	s.	d.	
Durban Rood. Deep	60	52,139	127,303	7	11	48	0	95,348	3	5	36	6	31,955	4	6	12	3	6 months
Rood. Central Deep	50	29,679	47,241	17	5	31	10	45,922	17	0	30	11	1,319	0	5		11	3 ,, *
Langlaagte Deep .	100	73,862	120 734	18	1	33	0	102,404	4	11	28	0	18,330	13	2	5	0	5 ,, *
Crown Deep . .	200	201,273	379,173	10	3	37	8	232,522	15	6	23	1	146,650	14	9	14	6	9 ,,
Robinson Deep .	100	116,121	314,561	0	0	58	10	198,158	4	5	34	5	142,008	0	0	24	5	12 ,,
Ferreira Deep .	60	28,160	88,984	14	1	63	0	44,924	18	0	31	10	44,060	0	1	31	3	3 ,, *
Village Main Reef.	100	87,279	177,764	0	0	63	6	117,087	19	4	26	10	160,618	0	0	36	10	6 ,,
Nourse Deep. .	100	136,480	292,185	7	4	42	9	205,133	2	2	30	6	84,052	5	2	12	3	12 ,,
Jumpers Deep .	100	159,462	338,004	6	7	42	4	232,449	19	9	29	1	105 554	6	10	13	2	12 ,,
Geldenhuis Deep	200	202,404	426,428	18	2	42	1	210,166	16	11	20	9	216,262	1	3	21	4	8 ,,
Rose Deep . .	200	230,100	485,553	5	7	42	2	232,908	5	6	21	11	232,645	0	1	20	2	8 ,,
Glen Deep . .	100	128,042	281,928	5	1	44	0	170,127	19	3	26	6	111,800	5	10	17	5	11 ,,

Average yield on basis of total tons and total yield equals 43·6 shillings per ton.
,, cost ,, ,, ,, ,, cost ,, 26·4 ,, ,,
,, profit ,, ,, ,, ,, profit ,, 17·2 ,, ,,

* Total time mills in operation.

CHAPTER V.

DIMENSIONS OF DEEP-LEVEL PROPERTIES.

It is generally held that a deep-level property should have an area of approximately 200 claims, this being figured as containing sufficient ore to keep 200 stamps running for 20 years. In the opinion of the author, this area is not sufficient for the third row of deeps, for the following reasons.

As a general statement, it is true that only two reefs are worked even in the best portions of the Rand—namely, South Reef and Main Reef Leader. Let us assume that the South Reef averages 4 feet in width throughout a given area of 200 claims, and the Main Reef Leader averages 3 feet in width in the same block. It is not probable that the angle of dip in the third row will average more than 20°.

At an angle of 20° and 4 feet width, the South Reef would yield roughly 20,000 tons per claim, and similarly, the Main Reef Leader, at 3 feet in width, would yield 15,000 tons per claim, or a total of 35,000 tons. From this total, the following deductions must be made:—

Total ore contained	35,000	tons per claim.
Deduct 10 per cent. for dykes and faults .	3,500	,, ,,
	31,500	
Deduct 20 per cent. for unpayable ore .	6,300	,, ,,
	25,200	
Deduct 20 per cent. for sorting, say .	5,200	,, ,,
Total milling ore . .	20,000	,, ,,

Now, a 200-stamp mill of latest pattern will crush, in a 28 working-day month, 28,000 tons of ore. The total estimated milling tonnage in the 200-claim block under discussion is 4,000,000 tons, which, divided by the monthly crushing, gives 143 months, or say 12 years only, as the life of the mine. It will probably be urged that in allowing 20 per cent. of the available tonnage for unpayable ground, after deducting for dykes and faults, I am taking too conservative a view. That, of course, is a matter of opinion and experience, and the writer would not care to allow less as a general rule. On the other hand, the assumption of two reefs throughout the property is quite a favourable one, and in many instances one that will not be realized; in fact, there are sections of the Rand in which the writer has only estimated a milling tonnage of 10,000 per claim.

On this estimate, to provide a property which at 20,000 milling tons per claim and a 200-stamp equipment will have a life of 20 years, we require 336 claims.

The boundaries of a property should be very carefully considered, in relation, firstly, to the dykes and faults in the vicinity —data of which are obtainable from the shallower mines—and also to the ore shoots.

In the case of the latter, the trend can be very accurately determined by a study of the assay plans of mines working nearer the outcrop in the same neighbourhood, and where large areas are being split up into subsidiary companies, this aspect of the question should receive every attention. It might easily happen, as pointed out elsewhere, that of two companies placed south of developed outcrop properties, a boundary-line, laid out on paper, might give to the one company all the best section of ground at the expense of the other.

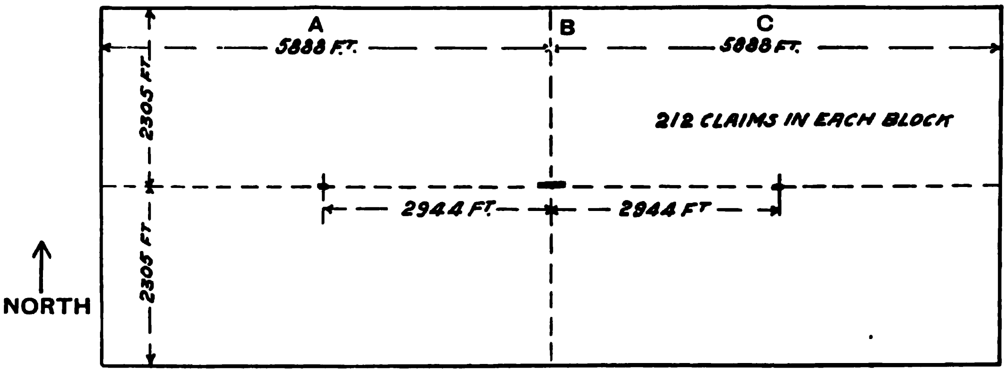
As a general scheme, deep-level properties should have, where possible, a dimension along the strike of three to four times that along the dip—the nearer the greater figure the better—for the reason that in developing such an area, and having encountered poor ore at the start, the great width on strike allows of the possibility that in continuing the work of driving, ore of better grade will be met with, whilst, were the property narrow along the strike, and the same conditions of poor ore encountered, the chances of opening up better grade rock are minimized directly as to the narrowness of the property in the strike direction.

CHAPTER VI.

LOCATION OF DEEP-LEVEL SHAFTS, AND BORING FOR REEF HORIZON.

The Location of Deep-level Shafts.

THE practice in the past, in the laying out of deep-level shafts, has been to choose positions as near the outcrop as the boundaries of the property would permit. Although, from the engineer's standpoint pure and simple, this method does not commend itself to the writer as being the best, for reasons given at length in a paper read before the South African Institute of Engineers in 1897, yet for financial reasons, which have to do with the realization of earning capacity upon the invested capital in the shortest time, there is no doubt that it is the best. In a recent paper* read before the American Institute of Mining Engineers, a departure from the former method has been made, by advocating and adopting a scheme of joint and combination shafts, which shall be common to several companies. The scheme is illustrated in Fig. 20.



SKETCH PLAN OF FOUR DEEP LEVEL PROPERTIES.

Fig. 20.

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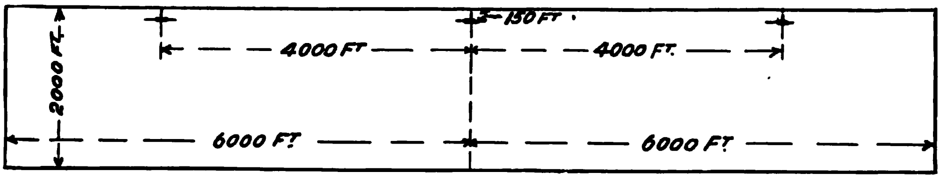
* "Deep-level Shafts on the Witwatersrand," by T. H. Leggett.

The four deep-level companies under this scheme agree to sink the three shafts, in the positions shown, the combination shaft being sunk by four participants, and the joint west and joint east shafts, respectively, by two participants. It is claimed that this method best serves the four companies concerned, for the development of their properties.

In comparing the methods of developing the properties, it has been taken for granted that the lower companies are in any case dependent upon the good offices of the upper companies for development, since the only comparisons drawn are as between the joint shafts, in the position shown, and as from shafts sunk at the points A, B, and C in the northern properties, and from the reef horizon in those shafts, by inclines through the total breadth of the said northern properties, until finally the southern companies' ground was reached.

If it were true that the only plan of attack for the southern companies lay in some form of co-operation with the northern companies, then the writer would heartily endorse the views expressed as to the advisability of using the central shaft system recommended, but the southern companies are by no means so dependent. They have the option, if they think fit, of sinking a joint shaft on their divisional boundaries, and just inside their northern boundary, and of sinking shafts quite independently in any other position chosen, or in the positions shown in Fig. 21.



SUGGESTED LOCATION OF DEEP LEVEL SHAFTS.

Fig. 21.

This is, in the writer's opinion, a better location than that advocated in the paper under notice, for the reason that each shaft commands an equal length of development, and therefore, all other things being equal, connection should be made between shaft and shaft, and the levels reach the lateral boundaries at the same time. Also the equal division of the total length avoids the long tramming distances from the outer sections, which the criticized location would involve.

The scheme advanced in the paper under notice is undoubtedly one which, from a financial aspect, favours the southern companies at the expense of the northern, and may be expressed in the following statement:—

"A" company sinks its shafts on its northern boundary, and begins milling operations four years after its formation. "B" company sinks its shafts in the centre of its property, and begins milling operations five years after its formation. What net advantage does "A" gain over "B" on account of entering the profit-earning stage 12 months before "B"? Assume for both cases—

(1) The properties and the tonnage contained to be equal.
(2) 200 stamps as the basis of equipment.
(3) Life of mine, 10 years.
(4) Annual profit, £144,000.
(5) Work started, January 1st, 1903.

"A" company would begin milling on January 1st, 1907, and it would have in hand 12 months' profits by January 1st, 1908, when "B" would begin.

By January 1st, 1917, "A" company would have earned £1,440,000 in profits. If the profits were annually deposited to bear 5 per cent. compound interest, the amount would be, at the date named, £1,731,825. (This is the amount of an annuity of £144,000 for 9 years bearing 5 per cent. compound interest, *plus* the last amount of £144,000 earned in the tenth year, upon which no interest would have accumulated.) The position of "B" company would at the same date be, that it had earned £1,296,000 in profits.

Treating these profits in the same manner as before, the amount is £1,519,071.

"A" company has exhausted its mine, but "B" company has still to earn £144,000. The present value of £144,000, due in 12 months, is £137,142; and this amount added to £1,519,071 reflects the value of "B" profits at January 1st, 1917, thus:

$$£1,519,071 + £137,142 = £1,656,213.$$

The difference in "A's" favour at the date named is therefore £75,612; which amount represents 5 per cent. compound interest on the profit made by "A" in its first year.

The present value of £75,612 due in 10 years is £46,409.

The *actual difference* is therefore simply the compounded interest on the first year's profits, and whether it be one month's or twelve months' difference, this is the basis upon which the calculation must be made.

As a matter of fact, in assuming a difference of 12 months in favour of the northern vertical as compared to the central vertical method in the above statement, an extreme figure has been taken, but for the requirements of the illustration it serves its purpose.

It is clear that, as interest is obtainable upon cash profits, the earlier these are in hand the greater will be the ultimate amount. In the comparison of two propositions, however, such as those above cited, it is equally clear that, as the actual amount to be netted in profit, exclusive of interest, is the same in either case, the only element of difference must lie in the factor of interest.

If, as stated in the concluding remarks, "it suggests the policy of a general pooling of interests in such a block of four companies, or their amalgamation into a single concern," it resolves itself into one large property of 848 claims in extent, which it is proposed to work by central vertical shafts, or a scheme similar to that suggested by the writer in 1897, but abandoned later because it delayed the mines' producing stage by some considerable time, which latter the writer of the said paper admits is, in deep-level mining, "the essence of the contract."

A comparison of the various methods suggests indubitably, in the writer's opinion, for each pair of deep-level companies in strike direction, the adoption of the principle of shafts sunk near the northern boundary, so that the producing stage may as early as possible be attained, and the principle of the joint shaft on the divisional boundary, for the reduction of expense in shaft-sinking, the positions of shafts being shown in diagram.

Boring for Reef Horizon.

When the boundaries of the deep-level properties are fixed, and the positions of the shafts are located—after study of the prevailing lines of dyke and fault—diamond drills should be brought into position adjacent to the shaft and work immediately begun, shaft-sinking commencing simultaneously. This is the course the writer has adopted at the Cinderella Deep, for the following reasons:—

(1) The diamond drill pierces the formation rapidly, and gives the engineer a perfect section of the rock through which the shaft has to pass.

(2) The presence of longitudinal faults which lie beyond the region of development in the upper adjoining mines is disclosed, and also the presence of dip faults, which may cause great vertical displacement in the reef's position between the two shafts.

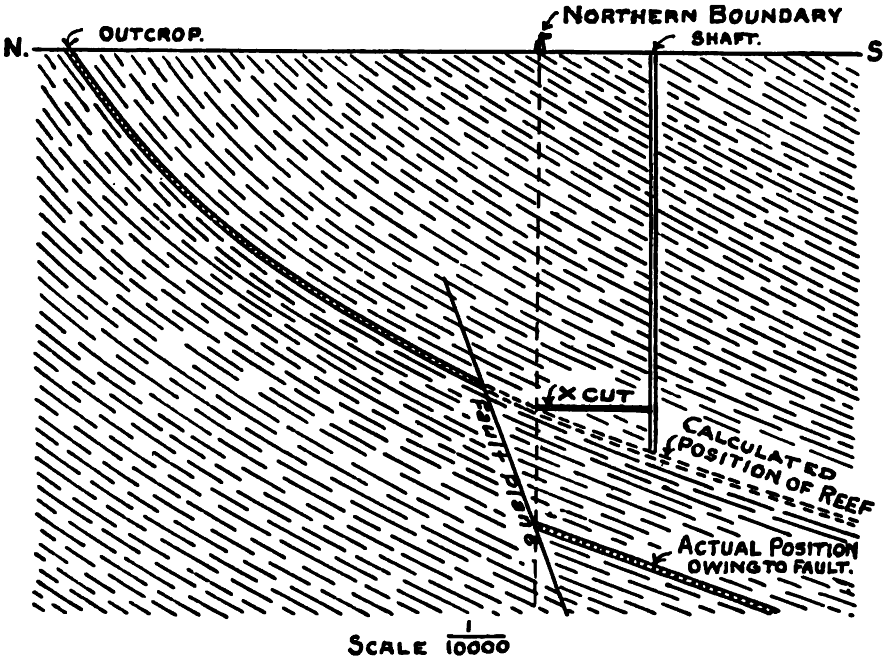
(3) The reef horizon is determined with certainty, thus giving opportunity for starting the first cross-cuts from the shaft to the reef. This is a most important point, as the sinking and cross-cutting go forward simultaneously from the horizon of the first cross-cut, and thus the reef is struck at about the same time in both cross-cut and shaft. This enables a connection to be made on the reef from the first level to the second in the earliest possible time, thus aiding ventilation of the workings, and also enabling a start to be made in the driving of both first and second levels at practically the same date.

In case no borehole has been cut, and the shaft is sunk until the reef is encountered, there is the decided loss of the time taken to sink from the point of first cross-cut to the reef, and the additional loss of time and the expense involved in starting the cross-cut from the timbered shaft.

It may also happen that a figure of depth is arrived at on seemingly certain data from the upper company's workings, and at a calculated depth the first cross-cut is started out to intersect the reef in the projected position, but, owing to unforeseen conditions, the reef may be displaced and the said cross-cut go to the northern boundary without having intersected the reef, thus proving nothing. This feature is illustrated in Fig. 22.

The cost of boring is, of course, an item of expenditure, since it would amount to probably not much less than £30,000 for two holes of 6000 feet each, but the information obtained is, firstly, of great value as a time-saver, and, secondly, obviates heavy and useless expenditure for cross-cutting; moreover, the value of the interest on the capital and dividends of the company, calculated on the saving of a clear three months, will be found to amount to a moiety of the sum originally spent in obtaining the information which resulted in the said saving of three months.

Finally, the diamond borehole gives much valuable information for use in the laying out of the pumping and hoisting equipment required on the shaft concerned.



SECTION TO ILLUSTRATE ADVISABILITY OF BORING TO LOCATE REEF POSITION.

Fig. 22.

The following noteworthy boreholes have proved the Main Reef horizon in the deep-levels :—

Position.	Approx. horizontal distance from outcrop.	Vertical depth to reef.	Approx. average dip.
Bezuidenville	5,800	3130	25 degrees
Geduld { No. 1 hole	9,500	1391	7 „
Geduld { No. 2 hole	14,500	1724	7 „
Geduld { No. 3 hole	16,000	2136	7 „
Turffontein { East hole	8,565	4887	20 „
Turffontein { West hole	8,575	Present depth 3006 feet. Reef not yet struck.	

The Locality Plan attached hereto (Plate V.) shows the relative position of the Bezuidenville and Turffontein holes in respect to the outcrop and the surrounding properties.

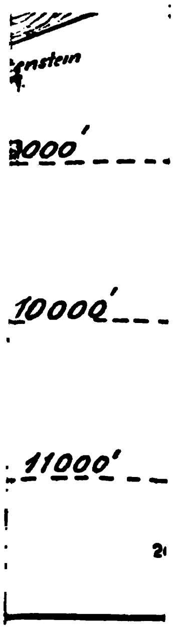


Plate V.

The section of the Bezuidenville borehole shown below in Fig. 23 has been compiled from information obtained from a paper by Mr. G. A. Chalmers, written in that connection.

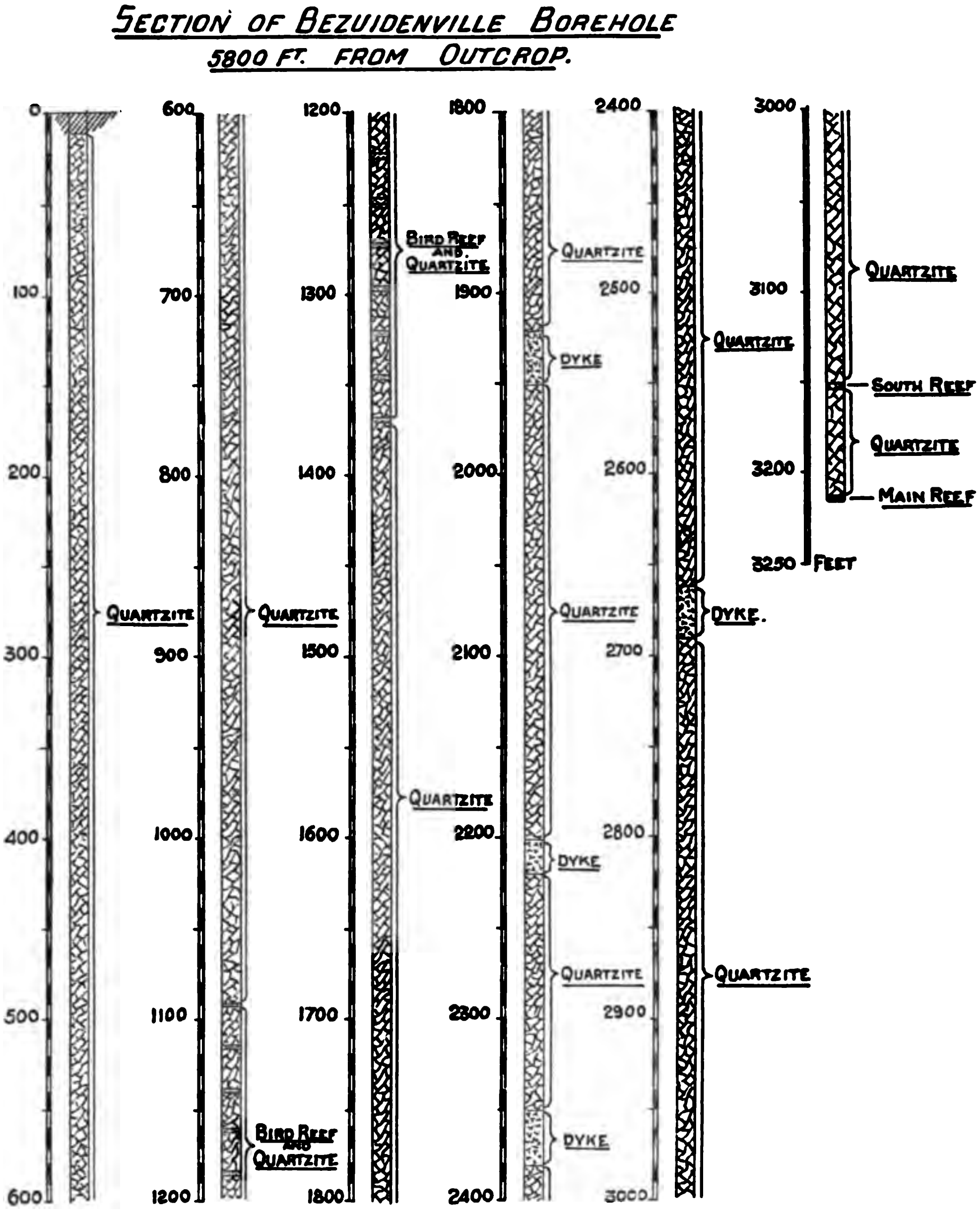


Fig. 23.

CHAPTER VII.

SURVEYING DEEP-LEVEL SHAFTS.

THE time-honoured methods of plumbing and surveying deep-level shafts must soon, in the writer's opinion, give place to more accurate and scientific means, in view of the great depth to which shafts will be sunk in the future.

The errors which are possible when dependence is placed upon a swinging plumb-bob, notwithstanding all precautions of winged plummets, mercury baths, etc., even at shallow depths, at once suggest themselves. How much greater the errors which would accrue if the method were adopted for shafts 3000 feet deep and upwards, and leaving out of consideration such niceties as the error in verticality of the plummet at these depths, owing to the difference in the rotative velocity of the earth at its surface, and that at, say, 4000 feet below.

To overcome the objections to the plumbing method raised by the writer, a method has been devised by Mr. P. B. Osborn, his Chief Surveyor, which is at once scientific, practical, and efficient. Mr. Osborn thus describes it:—

"The following method of accurately fixing points at the bottom of a vertical shaft of considerable depth, from which a subterraneous survey can be connected with the surface triangulation, obviates entirely the use of suspended plumb-bobs for an indication of verticality and direction.

"Referring to diagram (p. 47). The surface station B is first placed on the projection of a longitudinal line drawn through the point A and several other points fixed over the shaft compartments. The station B should be well defined, and should be placed at a greater distance from the shaft than the depth of the latter. The accurate vertical distance between the points A and B is now established by means of a good level.

"Having placed all the points on the surface accurately on one line, we set up a transit theodolite, especially constructed

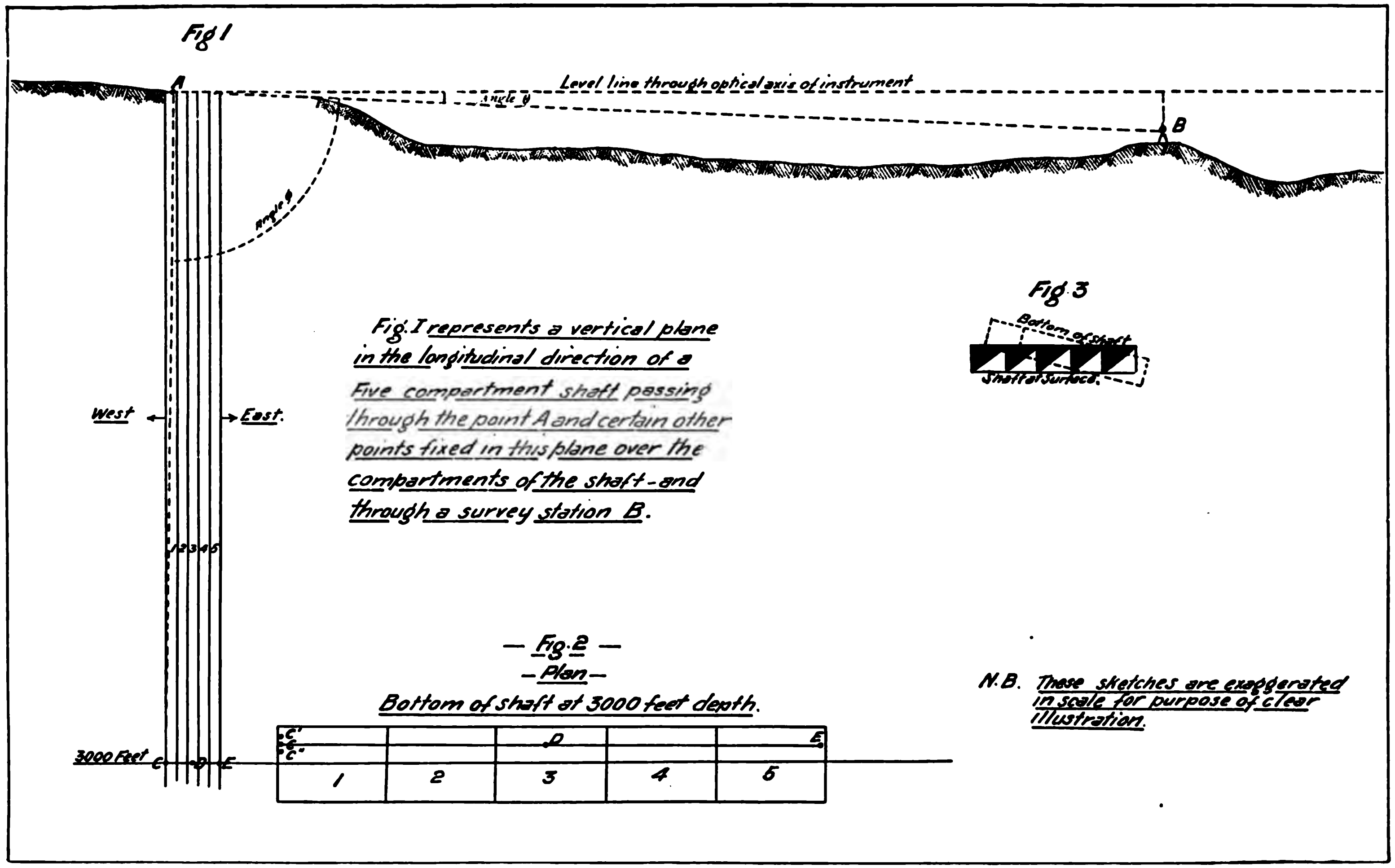


Fig. I represents a vertical plane in the longitudinal direction of a Five compartment shaft passing through the point A and certain other points fixed in this plane over the compartments of the shaft - and through a survey station B.

N.B. These sketches are exaggerated in scale for purpose of clear illustration.

Diagram illustrating Mr. Osborn's Method of Determining Points at Foot of Deep Shafts.

for this work, at the point A over the No. 1 compartment of the shaft. Then, by sighting on to B, and depressing the telescope in verticality, we proceed, by a series of careful observations, to establish any point C′, at the bottom of the shaft, as near to the west side as possible, and, by reversing in azimuth and altitude, we repeat a series of observations and establish a second point C″, which is placed by the assistant roughly at right angles to the line AB. From C′ to C″ will be the error of the instrument.

"If we now take a point C, which should be in the plane of the great circle passing through B, and the centre of the instrument at A, we can proceed, by a series of observations, to find the angle ϕ between B on the surface and C at the bottom of the shaft, both being in the same plane. We have already obtained the value of the angle θ from the surface measurements, and from the value of these two angles, ϕ and θ, may be deducted the value of the small angle at A, which is subtended by the distance, at the bottom of the shaft, between the established point C and the imaginary point indicating the prime vertical through A. With the depth of the shaft as base, and the above small angle, we are able to calculate the co-ordinates of C. Having established the point C in No. 1 compartment, we proceed to establish the two points D and E, or more points, in the plane of the great circle through A and B.

"In every case the two extreme compartments should, if possible, be used, and the illuminating of the points at the bottom of the shaft should be effected by suitable electric devices.

"To carry out the observations above described, and dealing with a shaft 3000 feet or more in depth, a specially constructed 8-inch transit theodolite of the concentric type would be required, having micrometer microscope attached to the eyepiece, and the vertical axis enlarged and perforated sufficiently to permit of vertical observation; fitted also with a very sensitive striding level, which should be of such a height above the instrument that it will be free from contact when the telescope is pointing down the shaft.

"With this instrument, and a careful system of observation—by which errors of standard level, collimation, eccentricity, and graduation are eliminated—no reasonable difficulty should be experienced in establishing the points C, D, and E, or more points lying in the plane of the great circle through A and B, and situated at the bottom of the shaft, within a limit of area of two seconds of arc.

"Accepting this as correct, it is possible, by applying corrections to the points thus established independently at the bottom of the shaft, to reduce the error to half the above, or to one second of arc, which would mean, in numerical value, approximately 167 inches. The latter figure is arrived at on the basis that one minute of arc subtends a displacement of 1 inch in 300 feet, or 10 inches in 3000 feet, and therefore one second of arc will be subtended by one-sixth inch, or 0·167 inch. Taking the length of the line CE at the bottom of a five-compartment shaft as being 30 feet, we have deduced from the above observations that we may have a limit in deflection of the line CE from the direction of the line AB of 0·167 inch.

"The influence of this error in our base of 30 feet on an underground traverse for a distance of 1 mile would be about 29 inches, or of 1 in 2000, which can generally be considered sufficiently accurate, in view of the fact that underground surveys are classed as good when checked to within these limits of error. It should be noted here that the angle ϕ would be sufficiently accurate if observed to within a limit of error of 10 seconds, as an error of a few inches in the direction of AB would be of no consequence when spread over the length of the subsequent underground survey. The greatest accuracy is, however, necessary in the establishment of the points at the bottom of the shaft in the plane of the great circle passing through the points A and B on the surface, as from these data the whole of the survey proceeds. The shaft itself could be a few inches out in every direction at the bottom, and it would make no practical difference.

"While the shaft is in process of sinking, it should be kept in line by being subjected to this method of observation, first at a depth of 500 feet, and subsequently at intervals of 500 feet; otherwise it is liable to assume a position at the bottom reflected in the sketch in Fig. 3 (p. 47).

"The particular advantages claimed for this method lie in the facilities it affords for performing accurate work with an instrument which is not in perfect adjustment, the errors of the instrument being eliminated by the system of observation.

"In order to overcome unfavourable atmospheric conditions on the surface, the work must be done at night."

CHAPTER VIII.

DEVELOPMENT OF DEEP-LEVEL MINES.

Cost of Development.

Happily for the Witwatersrand, its immense possibilities have attracted a class of capitalist who, for enterprise in the tackling of ventures requiring the sinking of sums counted in round millions sterling, for honesty of purpose, for free and explicit statement as to the position and potentiality of their undertakings, are certainly not excelled in any other department of finance.

It is to this fact, coupled with the position which the Rand has won for itself as a gold-producing district, that the great amount of public confidence in these Fields is to be assigned; and that confidence, being permanently won, enables the promoters of mining schemes to successfully launch upon the market enterprises demanding collectively the employment of such vast sums as (within the like area) cannot be paralleled in all the history of mining.

The development of the deep-level areas of the Rand evince a boldness in expenditure which, on any other formation but that of banket, would be regarded as simple madness. The average capital expenditure on the Rand mines deep-level companies of the first row stands at £590,000, the highest being £813,000 and the lowest £459,000.

In the deep-level mines of the future, say in those of the third row, having shafts 6000 feet in depth, the cost of development and equipment will be considerably increased, and the time occupied to reach the metalliferous zone, and before development proper upon that zone begins, will be over five years, to which another two or three years must be added for the time required to develop and equip the property with the requisite ore reduction and treatment plant, and other matters in connection therewith.

In calculating upon the future cost of deep-level work, we cannot fairly base our figures upon the average sum which the deep-level mines of the first row expended, to bring the properties to the producing stage. The conditions obtaining respecting cost of labour and materials when the bulk of that work was in process of accomplishment were very gradually but surely undergoing alteration for the better, and there can be no doubt that the future work will be accomplished for very much less money than was required for the same class of work in the past.

The differences which exist in the capital outlay on a third row deep-level, as compared to a first row, apart from the interest charges, lie entirely in the greater depth of shaft to be sunk, the heavier pumping and hoisting equipment, and possibly the provision for mechanical means of ventilation; otherwise the cases remain the same.

The schedule of costs of a 6000 feet deep-level undertaking would stand somewhat as under :—

	£	*s.*	*d.*
Two diamond drill holes at 6000 feet = 12,000 feet at 45*s.*	27,000	0	0
Two 5-compartment vertical shafts 12,000 feet at £40	480,000	0	0
Two 4-compartment incline shafts 1000 feet each = 2000 feet, including equipment and stations, at £25	50,000	0	0
Cost of development of mine for 200 stamps	160,000	0	0
Machinery and plant at surface, other than shaft equipment	200,000	0	0
Buildings, etc.	50,000	0	0
Reservoirs	5,000	0	0
Dams	15,000	0	0
Stores and materials for stock	10,000	0	0
	£997,000	0	0

In the amelioration of conditions which it is believed the establishment of the new political régime will witness, there is every reason to suppose that the costs scheduled above will be very materially reduced, and specially the cost of shaft sinking. The writer believes that the average cost of this work will be reduced by £5 per foot within the next two years, making a total difference on the above schedule of £70,000. A corresponding reduction in the cost of machinery and plant will occur, when the ruling high ocean freight and railway rates are reduced to prices more in keeping with those obtaining in other countries.

TABLE OF ESTIMATED COSTS FOR DEVELOPMENT AND EQUIPMENT OF DEEP-LEVEL COMPANIES AT FORMER RATES.

TWO VERTICAL SHAFTS POSTULATED FOR EACH COMPANY.

Depth of vertical shafts.	Cost of bringing property to producing stage.			Cost to develop and equip for 200-stamp mill.			Total cost to bring to the producing stage.		
	£	*s.*	*d.*	£	*s.*	*d.*	£	*s.*	*d.*
3000 feet	240,000	0	0	425,000	0	0	665,000	0	0
3500 „	280,000	0	0	425,000	0	0	705,000	0	0
4000 „	320,000	0	0	425,000	0	0	745,000	0	0
4500 „	360,000	0	0	425,000	0	0	785,000	0	0
5000 „	400,000	0	0	425,000	0	0	825,000	0	0
5500 „	440,000	0	0	425,000	0	0	865,000	0	0

Time required for Opening a Deep-level Mine.

The average time required for opening a deep-level mine on the first row, and bringing it to the producing stage, as disclosed by the records of the Rand mines, is 3½ years. The average rate of sinking has, however, largely increased in the more recent shafts, specially in those of the Consolidated Gold Fields group, two shafts of this group having been sunk an average depth of 145 feet per month, over a period of 12 months. This rate, however, may not be taken as an average of all shafts, for the simple reason that the conditions vary in respect of the quantity of water to be handled, the hardness and bedding of the quartzite layers, or the occurrence in the shaft line of hard, massive dyke bodies. In shafts up to 3000 feet in depth, and under normal conditions, the rate of sinking will probably not be less than 115 feet per month throughout the sinking. The time required, therefore, to sink the shaft to the reef is 2·2 years.

In shafts of a projected depth of 6000 feet the average rate for the last 3000 feet of the sinking will be somewhat reduced, and the writer would not figure upon a greater average than 90 feet per month. The time occupied in sinking the total depth of 6000 feet will therefore be five years.

At this point development of the ore begins, and it may be assumed that the reef has been struck in a cross-cut from the shaft at about the same time that the shaft strikes the reef, thus making it possible to start out two levels at the same time.

The incline shafts will also probably be placed near the foot of the vertical—in cases where two verticals are employed—in order to facilitate handling the ore, so that these shafts will be started at about the same time as the first levels. The average rate of sinking these will probably be about 100 feet per month, so that, allowing some time for preliminaries—say at the end of 2½ months—the depth of the first level in the incline shaft is reached, giving two working faces from each shaft, or four in all. The cutting of the station will occupy another four weeks, and therefore the second level from this shaft will not be reached for five months from the time of commencement, the third at the end of 7½ months, the fourth at the end of 10 months, and the fifth at say the end of 12 months.

At the end of the period named the possible working faces would be as under :—

Two each from 1st cross-cut from vertical shaft	4
„ on main gallery at foot of vertical shaft	4
„ „ 1st level from incline shaft	4
„ „ 2nd „ „ „	4
„ „ 3rd „ „ „	4
„ „ 4th „ „ „	4

During the first 2½ months period, 4 faces are working, at an average rate of advancement of 80 feet	800	feet
During the second period of 2½ months, 8 faces are working, at 80 feet per month advancement	1600	„
During the third period of 2½ months, 12 faces are working, at 80 feet per month advancement	2400	„
During the fourth period of 2½ months, 16 faces are working, at 80 feet per month advancement	3200	„
During the fifth period of 2½ months, 20 faces are working, at 80 feet per month advancement	4000	„
	12,000	„

Whilst the driving was advancing, the putting in of the necessary winze connections would be accomplished.

The total of 12,000 feet of driving, on an assumed width of 4 feet, and backs of 150 feet, would develop 600,000 tons of ore. Of this total we must make the following deductions :—

Total ore developed	600,000	tons
Deduct 10 per cent. for loss by dykes and faults	60,000	„
	540,000	„
Deduct 30 per cent. for unpayable ground	172,000	„
	368,000	„
Deduct 20 per cent. for sorting	73,000	„
	295,000	„

The ultimate equipment of a deep-level mine is usually based on a milling capacity of 200 stamps. The ore developed in one year, as shown above, would only be equal to 10 months' development in advance of the mill, which would not be sufficient, as the number of stopes that would be available on such a development would not produce a supply of ore adequate to feed 200 stamps daily. It would be necessary, therefore, to develop for say one more year, in order to have sufficient ore for feeding 200 stamps. But here an important issue, mentioned in another paragraph, arises. If the mine be kept at the development stage for two years, the result is, firstly, that the shareholders are delayed one year in getting a return on the invested capital; and, secondly, the cost per ton of the ore developed during the second year is double what it would be were the mill running. These facts point inevitably to the conclusion that at the end of the first year the mine should enter the producing stage by starting milling with 100 stamps, for the supply of which there would be an ample development of ore, increasing the number of stamps from time to time as the development warranted.

It has been shown that the time required to sink a 6000-feet shaft is 5·7 years, and it has been further calculated that one year after the shafts had reached the reefs—always assuming no great dislocations interfering—100 stamps could be started; in other words, a deep-level property, in the position requiring 6000-feet vertical shafts, will reach the producing stage six to seven years after the commencement of operations, and will be running with its full equipment of 200 stamps seven to eight years after the same date. The following table shows the estimated time required to bring deep-level mines of various depths to the producing stage:—

Depth of vertical shafts.	Time required to bring to producing stage.
3000 feet	3·5 to 4·0 years
4000 „	4·0 to 4·5 „
5000 „	4·9 to 5·5 „
6000 „	6·0 to 7·0 „

Scheme of Development for Deep-level Mines.

The development proper of the ore in the deep-level mines begins when the vertical shafts have reached the reef.

The custom in vogue in the first row of deep levels, of turning the vertical shaft by means of a curve into an incline shaft, at the reef horizon, is one that will not be continued in the deeper mines. In these latter instances, the vertical shaft will have fulfilled its functions at the said reef horizon, and will thenceforth be the medium of hoisting and lowering all men, ore, and materials, besides forming the air and water ways from the mine.

Having the vertical shafts finished, and always assuming that they are located near the north boundary, the work of development proper will be commenced by starting out a main gallery, which will connect between the shafts, and run out to the east and west boundaries of the property. This gallery will be of large cross-section, to permit of free haulage facilities, and ample ventilation. At selected points between the vertical shafts, two or more main incline shafts will be sunk, of not less than 21 feet by 5 inside timbers, and divided into four compartments, two for hoisting ore, one for use as a sinking way, and the fourth for pump and ladder way. These shafts will be equipped with powerful electric or compressed-air hoists, placed in chambers above the gallery, and in such a position that ore hoisted through the shaft can be automatically dumped into a bin. Connections will be made from these shafts to one another, and the whole property developed in the ordinary way.

The arrangements explained above are illustrated in the annexed diagrams, Figs. 24 to 28.

In the development of a mine there are certain considerations of prime importance which must regulate the scheme to be adopted, chief of which are the following:—

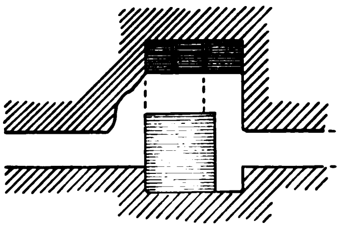
(1) The number of stamps to be fed.

(2) The rate at which ore can be developed.

(3) The relation between footage driven, and payable ore developed.

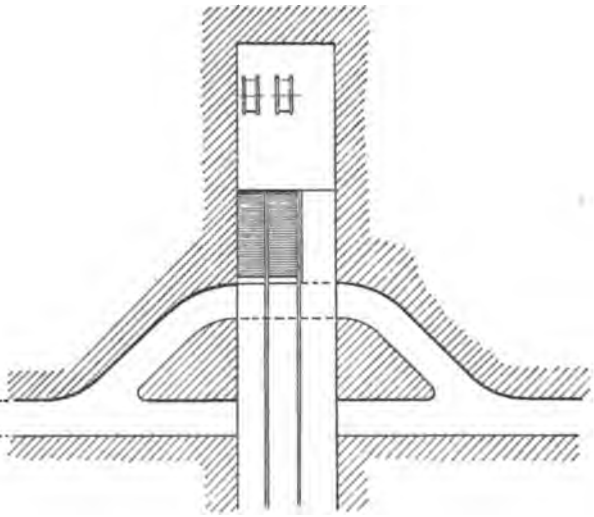
Assuming at the outset that the conditions of the mine are sufficiently well known, from the experience of neighbouring properties, to assess the number of stamps which the mine can conveniently employ, it has to be determined at what rate milling ore can be opened.

This involves a knowledge of the average conditions governing the ore shoots in the mine, which can only be definitely known by extensive development, between the lateral limits of the mine. A long level from east to west will, in normal circumstances, sample, as it were, the whole area of immediate development, and a fairly accurate estimate of the average ore which may be expected can be thus determined. If the said level were to show that the payable stretches of ore were very narrow, then the question of the cost of developing the small payable areas becomes a very serious one. Take an example. Supposing a shaft situated 2000 feet from a side boundary-line, and on a level driven from that shaft towards the said boundary, only 500 feet of payable ore were encountered, the said shoot of payable ore, lying adjacent to the boundary. In order to attack the ore in the indicated position, say at least 1500 feet of level would need to be driven. Assume that the cost to drive the level is £6 per foot—which it would be whilst the mine was in the developing stage—the total cost of the drive is £9000, to which add the cost of one winze, say 150 feet at £8, equal to £1200, and the total amount expended is £10,200.



PLAN OF STATION . VERTICAL SHAFT.

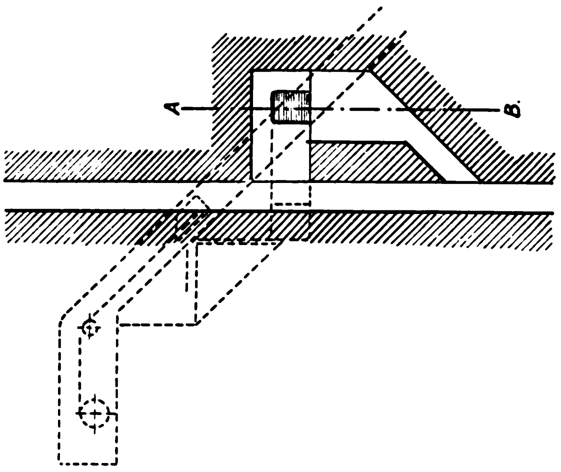
Fig. 24.



PLAN OF STATION AT TOP OF INCLINE SHAFT.

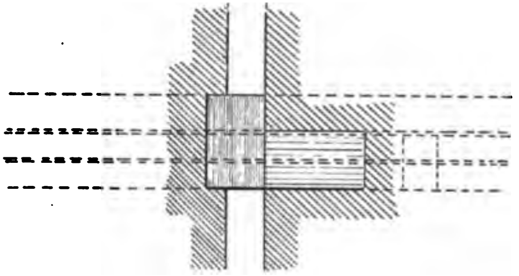
Fig. 25.

Now assume that the reef averaged 4 feet in thickness, and after deductions for sorting, etc., an average thickness of 3 feet was available for milling. The contents of an average block would be 500 feet (the length of shoot) by 150 feet (the height of backs) by 3 feet (the milling width). This gives a total of



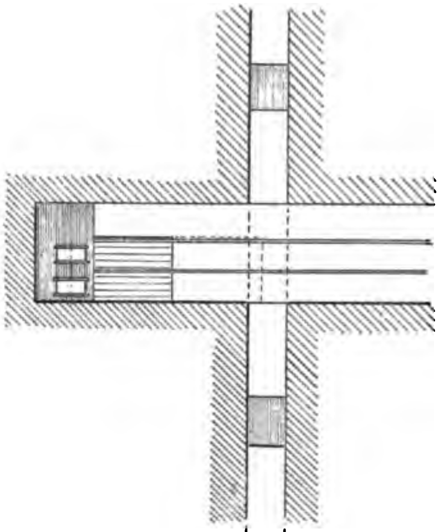
SECTION THROUGH VERTICAL SHAFT AND BIN.

Fig. 26.



VERTICAL SECTION ON A.B.

Fig. 27.



SECTION ON PLANE OF INCLINE SHAFT.

Fig. 28.

225,000 cubic feet of ore, which, at 12 cubic feet per ton, equals 18,750 tons.

The total cost to develop this tonnage we have seen to be £10,200, or at the rate of 10·8*s*. per ton. In figuring out the working costs, the sum of 10·8*s*. would have to be added to the operating charges, under the head of "mine development redemption." The ore would, therefore, need to be a high-grade quality to support this heavy charge, and make a profitable return. Of course the shoot might be separately developed by a central incline, but that would necessitate the sinking of two other winzes, one on either side of the incline, if the latter were sunk on the reef, and at least one if off the reef, whilst a separate hoisting installation would have to be provided, and the cost of these, and additional handling, figured into the calculation, would leave but little to recommend its adoption.

If instead of yielding 3 feet of milling ore, the block above figured upon were to yield only 2 feet, then the cost per ton for development would proportionately increase.

The object to be kept in view, once the ore shoots are located, is the disposal of the working places so that the greatest number of points of attack, consistent with a properly proportioned system of development, are obtained. It would not do, for instance, to place levels at intervals of only 30 feet, for the sole purpose of securing more points of attack; these must be placed at such intervals that the work performed from them will result in opening up a tonnage in proportion to the amount expended.

In the early stages of the development of a deep-level mine, it is all-important, other things being duly regarded, to plan the work so that the mine may, at the earliest possible moment, for reasons which appear elsewhere, arrive at the crushing stage. The first levels will, therefore, be laid off at intervals of say 150 feet, and the winzes at intervals of 300 feet or thereabouts, dividing the mine into blocks of 300 feet by 150 feet for stoping purposes. Each winze connection should give the starting-point for a double stope, and, depending on the width of the reef, it is easy to estimate how many stopes are required to keep the milling plant fully supplied. The stopes should be so arranged that they give a fair proportion of ore of payable grade from the various sections of the mine. As the mine progresses and development work gains on the milling requirement, the intervals between levels might with advantage be extended to 300 feet, and between winzes to 600 feet. Self-acting inclines will be used extensively for shifting the broken ore from the stopes to the levels, and therefore the need of handling in the ordinary way would be obviated.

The question of what tonnage development is necessary before starting a given milling equipment is one requiring nice adjustment. On the one hand, there is the undoubted advantage of having a very large development in order to enable the mine to return average yields, and at the same time to put it in such a position that, should large dyke and fault disturbances be encountered, the development in hand is sufficient to give time to exploit the disturbed areas, and regain zones free from their influences. If the mine be not developed well in advance of the mining requirement, the striking of a disturbed zone, such as that referred to, might ruin, at least temporarily, a fine property.

But there is another aspect of this question which is also important—namely, the relative cost of developing ore before crushing begins, and the cost when the mill is in full swing.

It may be taken as a general statement that all development costs are double before milling what they are subsequent to that stage being reached. This is accounted for by the fact that before milling begins the total costs of the administration are charged against the development work, and, when milling, all departments of the work—the mining, milling, and tailings treatment departments—bear a proportionate share of all general charges, so that the costs on development account are reduced by half.

The bearing of this on the matter in hand is obviously this. All ore developed before milling begins must be redeemed by a charge per ton double that which is made after that stage is reached, and therefore the working costs per ton will be higher by just the additional development charge.

The adjustment required in this matter is, therefore, such as, whilst giving a tonnage development reasonably in advance of the milling requirement, will not be excessively so, as all ore developed in excess means a constant, but unnecessary, extra charge to working cost, and, it follows, a reduction of profit.

The rate at which ore can be developed is very largely a matter of conditions, rock hardness, thickness of reef, and facilities for attack. The variations in rock hardness between mines under the control of the financial group which the writer represents are very striking. This can best be illustrated by the contract prices of headings in say the Roodepoort United Main Reef mine and the Aurora West mine. In the former the average cost of driving headings is 60*s*. per foot, and in the latter 105*s*. per foot, or 45*s*. per foot higher. The average performance in the first case for a contracting pair is 165 feet, and in the latter 90 feet. (The contractor in each case finds all labour, blasting material, candles, etc., the Company supplying rock-drilling machine, drills, and power.)

Again, the tonnage development is strictly proportional to the reefs that will pay for working and their widths. In some mines of the Rand the three reefs of the Main Reef series are payable—namely, Main Reef, Main Reef Leader, and South Reef. In some instances the Main Reef and Main Reef Leader are separated by such a thin layer of quartzite that they can be developed by driving only one level, which is generally on the Main Reef. In this case the Main Reef Leader is developed by making cross-cuts, generally at right angles to the bedding planes, from reef to reef, and from the said cross-cuts putting in the development "rises" in the ordinary way.

In other mines all three reefs require separate drives. In such conditions as these it is obvious that development will be rapid, since the three drives are virtually taken from the same attacking point.

In the instance of only one reef, and that narrow, each point of attack, say from a shaft, cross-cut, or winze, only gives two working faces, in contrast to the case above mentioned, which would give six.

The average monthly driving, by machine rock-drill contractors on the Rand, is about 80 feet. It is, however, not usual to get a level advanced at this rate monthly. The general system is to let to the contractor two separate working places, it being arranged that, whilst drilling is going on at the one face, the broken rock from the other is being cleared away in preparation for the next shift, and so on alternately.

If it be assumed that the "backs" from the level under process of driving are 150 feet, and the thickness of reef 3 feet, the ore developed for every foot of driving will be 37·5 tons. If such a mine, and with only one payable reef, had to supply a 200-stamp mill, driving would have to proceed at the rate of at least 1500 feet per month, and 750 feet of winzing would be necessary—when making due allowances for sorting and unpayable ground—to meet the requirement. To do this work would necessitate the constant running of 30 drills on development.

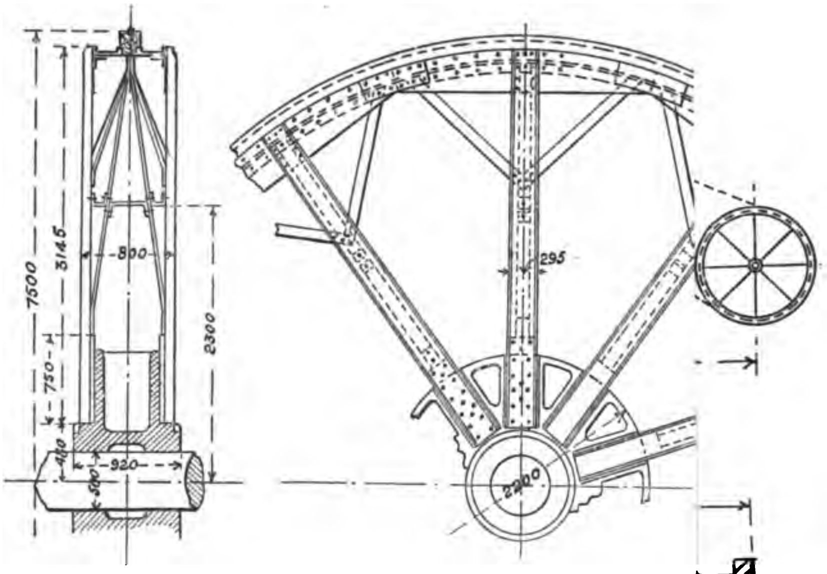
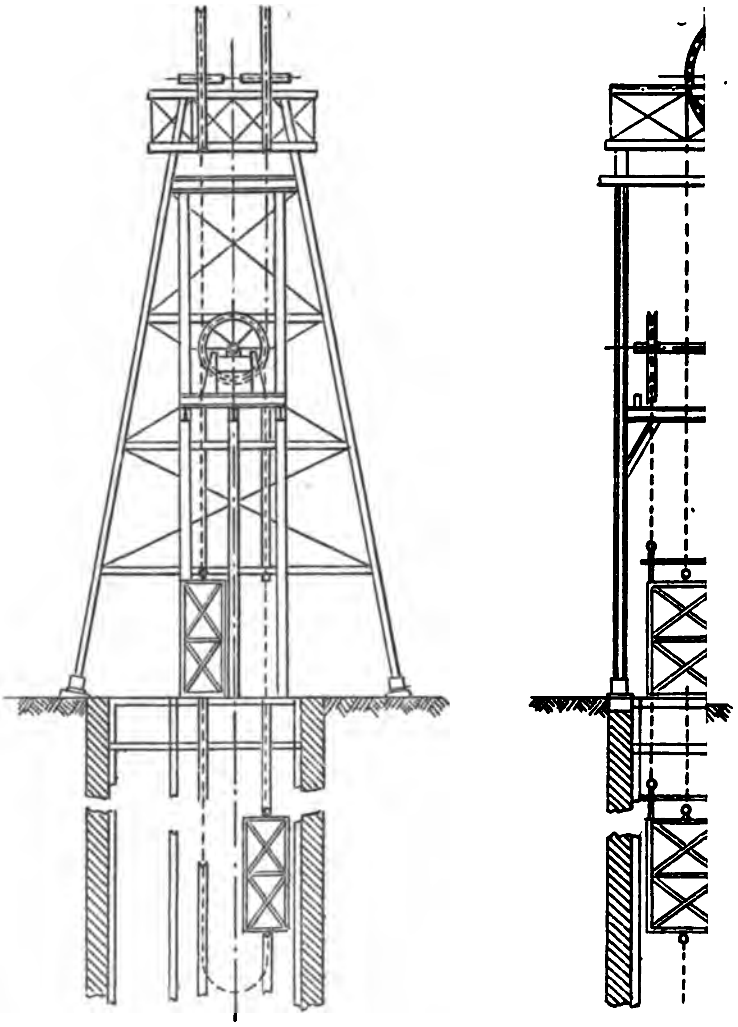


Fig. 29.—Winding Pulley.



To assist the engine in overcoming the period of inertia, the drums are built conical in shape and spirally grooved, the first lift being taken on the small end of the cone, where the rope speed is lowest, and gradually increasing as the load ascends the shaft. The objection to this design is, that in order to take the length of rope necessary for a 4000-feet lift, the drums would be excessively long, and hence the rope would "angle" to a dangerous degree on the headgear pulleys.

It is not improbable that the deep-winding engine of the future will be fitted with drums which are a combination of the conical and cylindrical, that is, conical for the first and cylindrical for the other half, the size of the drums being so gauged that an overlap of rope on the cylindrical portion is obtained, though this latter feature, having due regard to the life of rope, should, if possible, be avoided. An engine of this class has recently been built for the Tamarack Mining Company, of the following dimensions. The drum is 25 feet in diameter and 24 feet 6 inches in length, in shape partly conical and partly cylindrical, having a capacity for 6000 feet of 1½-inch rope, and a weight of 300,000 lbs. The drum is operated by four steam cylinders, each 34 inches by 60 inches.

The method used in Germany for deep winding is what is known as the Koepe system, which is illustrated in the accompanying outline drawings—Figs. 29, 30, 31, Plate VI.—reproduced by the writer from the *Engineer*. In this system, as in the Whiting system, only one winding rope is used, but instead of the drums of the Whiting hoist, with about three turns of rope round each, the Koepe employs only one grooved winding pulley. The rope passes from the one cage over the headgear pulley and round the driving pulley on the under side, returning on the top side again to the second headgear pulley, and thence to the second cage; a balancing rope extending from the bottom of the one cage to that of the other, passing round a groove pulley at the bottom of the shaft.

The disadvantages of this system are obviously the liability of the rope to slip in the groove of the driving pulley, more especially when starting the cages from rest, and the inertia of the load has to be overcome. These difficulties, however, have been minimized by the adoption of a special form of groove pulley, the rope running either in a hornbeam or on a rope packing. It is significant, however, that, notwithstanding the cheapness of this system both in installation and running, in the most recent instances of deep-level equipment in the United States—where the tail-rope systems have had abundant trial—the part conical, part cylindrical, drum has been adopted, although necessitating machinery of enormous proportions to do the required work.

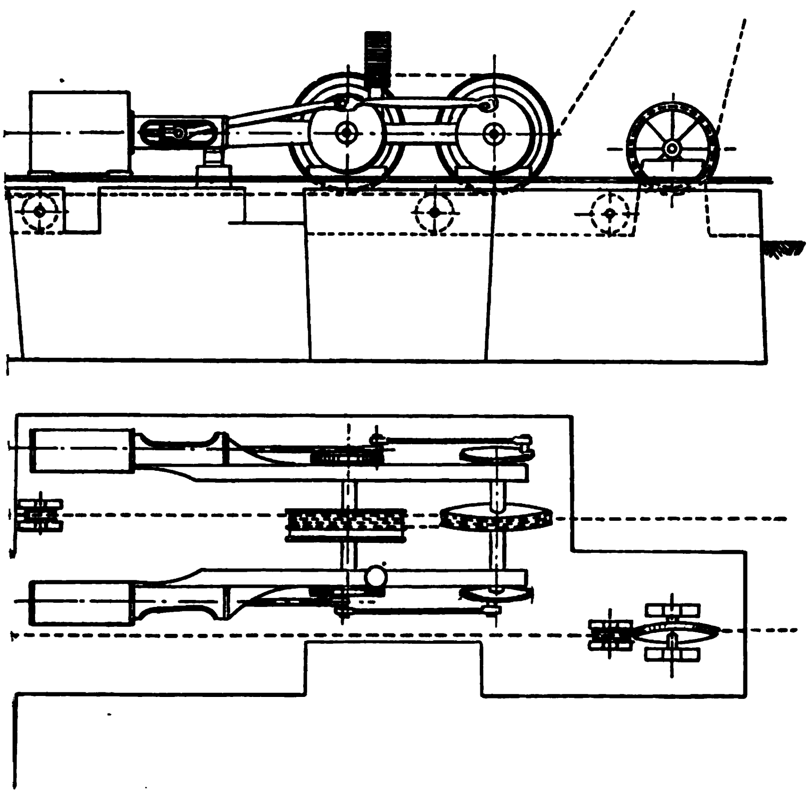
The engines now building for the heavy requirements of deep-level work are generally of the cross-compound condensing type, with steam-operated reversers, steam brakes, steam clutches, indicating at full load from 900 to 1200 horse-power.

A similar form of hoist to the Koepe, which finds favour amongst some engineers, is that known as the Whiting, illustrated in the drawings herewith (Figs. 32, 33, Plate VII.), for which the writer is likewise indebted to the *Engineer*. It consists substantially of two grooved drums, arranged in line. The winding rope from one side of a double hoistway is wound over both drums, a certain number of turns, generally three, and from thence taken back and down the second hoistway. If it is desired to wind from varying depths, then a form of take-up is adopted, which consists of a sheave on a movable truck, running out along the surface. In the case of employing this take-up, the rope, after passing over the drums, goes round the sheave on the movable carriage, and from thence to the headgear pulley, and down the shaft. The length of run of the carriage must be half of the greatest depth from which it is intended to hoist.

Following are some particulars of a winding plant of this description, winding from a deep-level inclined shaft in the United States:—

Depth of shaft	5000	feet.
Diameter of rope	1¼	inches.
No. of turns of rope round drum	3	
Revolutions per minute	90	
Speed of skip in shaft per minute	2000	feet.

The engines are directly connected to the back drum and to the forward drum—which is inclined to the back one by the width of one groove—by parallel connecting rods. The engines are a pair of tandem compounds with Corliss valves, high-pressure cylinder of 16 inches diameter, and low-pressure cylinders of 32 inches in diameter, the stroke being 48 inches. The exhaust steam is led to a condenser, of the jet



ding.

[*To face p.* 64.

type. A foot brake is used, and steam reversing gear. The drum being small, the engine gets up to full speed in three or four revolutions, and steam is carried nearly to the end of the run. It works with extreme sensitiveness and regularity.

Direct-winding Drums.

A new arrangement of direct-winding drums has recently been patented, by which drums of any length may be used without any fear of the rope overlapping or angling. It was thus described in the *Engineer*:—

Messrs. Marshall and Hopwood are the inventors. They place the axis of the winding drum or drums in a position at right angles to the headgear pulleys, and horizontal guide pulleys are provided to direct the winding rope to and from the headgear pulleys; each of these pulleys has a groove in a plane tangential to the periphery of the drum or drums. The guide pulleys are mounted on carriages running on rails fixed parallel to the axis of the drums, and by suitable gearing the guide pulleys and carriages are made to travel on the tracks in front of the drums at a rate controlled by the revolutions of the drums, so that the rope is taken off from, or fed on to the drum or drums at proper intervals, and when passing off or on the drums is caused by the guide pulleys to travel in a direct line to or from the headgear pulleys.

The drums are mounted on the same shaft, so that in turning in the same direction one winds the rope on while the other unwinds it; the guide pulleys and trucks are accordingly arranged at different levels, and are drawn along by nut-and-screw spindles driven by gear from the drum-shaft in such a way that when one moves forward the other moves backward to a corresponding extent, but both guide the rope straight in the direction of the headgear pulleys, either from the headgear pulley and rope wheel on to the drum so as to prevent any overlapping, or from the drum to the rope wheel and headgear pulley so as to prevent angling.

In winding, the following is the course of the rope: it bends vertically over the headgear pulley and again vertically under the rope wheel, then horizontally partly round the guiding pulley, finally, vertically on the drum, being subjected to the same flexures in the reverse order during unwinding. Many mining engineers disapprove of the manifold bendings of the rope incident to this, as to the Whiting and similar systems. Two drums are not necessary in the Marshall and Hopwood arrangement, but one drum may be used with one rope winding on whilst the other unwinds, and the guide pulleys and trucks may be traversed either by a nut-and-screw spindle, or by endless chain or other gear from the drum—the mode of traversing, in fact, would be adapted to suit conditions. For a plant designed to wind 4½ tons 6000 feet, Messrs. Marshall and Hopwood (we are informed by Mr. Marshall) intend using a drum 14 feet in diameter and 31 feet long, and a rope 2¼ inches in diameter, with safety factor taken at 7; the rate of winding is to be 30 feet a second, and the estimated horse power required is 1300 indicated. The guide pulleys will be 9 feet in diameter, and the track 4 feet gauge, the traversing is to be effected by a haulage rope and pulleys worked by an inclined shaft and bevel gear from the drum shaft.

As a means of overcoming the weight of rope in suspension at great depths, engineers are now specifying for taper rope, the taper calculated on the gradually decreasing load sustained by the rope as it ascends the shaft.

For the very deep shafts now projected on these Fields—6000 feet and thereabouts—and to hoist from these the heavy skips of ore that will be necessary for keeping a big mill supplied, the winding will probably be done in stages. In the case of a shaft 6000 feet in depth, the first landing would probably be arranged at 3500 feet from the surface, and at this position an engine would be placed for hoisting from the bottom of the shaft, dumping the ore automatically into bins which would feed into the first stage skips by shoots in the ordinary way.

The arrangement of the winding engine will probably be, for the first stage, a powerful steam engine, and for the second stage an electrically- or air-operated hoist. In either event it is patent that the cost of hoisting will be quite considerably increased.

The problem also remains to be solved by actual experience as to which method will give the most economical results consistent with the object to be achieved—namely, the landing of a given quantity of ore on surface in the shortest time and at the cheapest rate.

Hoisting and Lowering Labour at the Mines.

The next problem before the engineer is that of the best means of introducing labour into the deep-level mines. This is a feature which will tend to demonstrate the difference of deep-level mining as against outcrop mining in no uncertain way. In an outcrop mine the whole of the native labourers find their way to their working places in the mine by arranged footways, and, at the conclusion of their "shift," or task work, as the case may be, they return to the surface by the same means. Their ingress and egress is, therefore, not a calculable factor in cost to the company.

In a deep-level mine having a 6000-feet vertical shaft, however, the total labour force will of necessity be lowered and raised by some means which will call for the consumption of power to be provided at the cost of the company. The lowering of each "shift" of the force necessary to operate a mine feeding 200 stamps, will occupy a winding engine (assuming that three deck cages are swung, and arrangements made for simultaneous entrance and egress on the separate decks, and that each deck will carry 15 individuals) a space of 9 hours, or with two engines about 4½ hours each.

It is plain that, in these conditions, there are problems of (1) the most advantageous dividing up of the mine for ventilation and blasting, so that the latter can take place independently in various sections, at different times, without interfering with work in other sections; (2) the arrangement of the hours for working given stopes; (3) the arrangement of the underground work generally, so that, as far as possible, the cages will be running in balance. All these are matters for which only the experience of the case will indicate the proper adjustments. Doubtless some steps will be taken in the direction of an apparatus running constantly in the shaft, in the form of an endless lift, which could be utilized for the conveyance of the native labourers.

Ventilation.

The ventilation of the deep-level mines has been regarded in some quarters as a very knotty problem for the engineer to solve. The grounds for this opinion are accumulated from data obtained in mines or districts in which wholly different conditions prevail from those obtaining on the Rand. In general, mining districts are characterized by a certain number of isolated shoots or chimneys of ore, and between the said shoots or chimneys the distance is usually too great to admit of connection from one set of workings to the next. On the Rand, *per contra*, the whole length of reef in the pay sections is mined, and each company at some time or other makes connection to its neighbour. Moreover, the regular laying out of levels, winzes, etc., and connections through not less than two shafts to surface, induces such a strong natural draught that, in all main workings, very good ventilation is secured. Levels have been run, as at the Crown Deep, at a vertical depth of 1000 feet, for a distance of 1800 feet, by air drills, without any ventilating connection. The general practice, however, is to ventilate at distances of from 300 to 500 feet, these being the intervals for winzes, and therefore the level connections.

It is quite easy to understand that in districts where the rocks are heated, and the mining work is circumscribed by the limits of the pay ore shoot, the temperature of the lower workings becomes a serious drawback to development. In the case of the Rand, however, there exist the long levels at regular intervals in any given mine, which have as storage reservoirs the lenticular spaces formerly occupied by reef, the levels themselves forming ventilating channels, drawing on the air at every section of the mine which is in connection. These openings must also have a cooling effect upon the temperature of the rocks themselves for some little distance at least from the presented surfaces, and so an all-round cooling effect is obtained.

Certain experiments were carried out in 1896 by Mr. M. Franke, a then inspector of mines, in some of the deep-level shafts, to arrive at a figure somewhat approximating truth which would express an average vertical depth for each 1 degree centigrade rise in temperature. The figures obtained were as follows :—

Shaft.	Temperature about 50 feet below the mouth of shaft.	Temperature at bottom of shaft.	Vertical depth to bottom of shaft.	Difference in temperature.	Difference in depth.	Depth for every one degree centg. rise in temperature.	
	Degrees Centigrade.		Feet.	Degrees Centg.	Feet.	Feet.	
Langlaagte Deep, No. 1 shaft .	18·3	22·2	1342	3·9	1292	331	These shafts were not connected.
Langlaagte Deep, No. 2 shaft .	17·7	21·1	1099	3·4	1050	308	
Crown Deep, No. 1 shaft .	17·0	21·0	1352	4·0	1302	325	This shaft was connected.
Geldenhuis Deep, No. 1 shaft .	21·1	22·2	941	1·1	892	810	These shafts were connected.
Geldenhuis Deep, No. 2 shaft .	16·7	18·9	928	2·2	879	400	
Rose Deep, No. 1 shaft .	24·4	26·1	1201	1·7	1151	676	These shafts were connected.
Rose Deep, No. 2 shaft .	24·4	26·1	1099	1·7	1050	616	

It will be observed that there are enormous differences in the average depth for every one degree increase in temperature.

The shortest interval is obtained from the Langlaagte Deep, in which case, however, the shafts were not connected. The Rose Deep gives approximately the same interval from both shafts, whilst the Geldenhuis Deep gives double the interval in the No. 2 shaft of that obtained from No. 1 shaft, the latter being obviously at the time of taking the temperatures the downcast shaft. The writer is not, however, aware under what conditions these experiments were made. If the temperature in an upcast shaft were taken soon after blasting in the development ends and the stopes, there would naturally be an increase of temperature owing to the absorption by the atmosphere of the highly heated gaseous products of combustion, arising from the dynamite and gelatine explosions.

We may suppose that the experiments on the Rose Deep were carried out under normal conditions, inasmuch as both shafts show identical results in the recorded temperatures. If such a figure could be accepted—a rise of temperature through 1 degree centigrade for every 600 feet vertical depth: and the average temperatures obtained from the two Geldenhuis Deep shafts rather support this—then we should only have, in well-developed deep-level mines, a rise of 10 degrees centigrade in temperature above the average temperature of the shafts at, say, 50 feet from surface.

It is probable, however, that in taking this figure we are using a factor which is too favourable, and an average of the lowest results from the connected shafts may be taken as indicating the true factor. This figure would be 450 feet nearly. The average temperature of shafts and ventilated workings on the Witwatersrand deep-level mines we know does not exceed 24 degrees centigrade, and assuming a vertical limit of 200 feet at this temperature, and an increase of 1 degree centigrade for each 450 feet thenceforward, the temperature at 6000 feet vertical would be 33 degrees centigrade, equivalent to 91 degrees fahrenheit. This temperature, there is no doubt in the mind of the writer, would gradually be lowered as development proceeded, and could be directly reduced by the introduction of mechanical systems of ventilation, which up to present limits have not been required.

The conclusion to which these considerations lead is, that it is improbable that mines working a vertical depth of 6000 feet will suffer from any extraordinary heat conditions, or from such a temperature that the ordinary routine of the mine working and development will be affected.

Pumping.

The experience of these Fields regarding water in the mines —with one marked exception, that of the Connor shaft, on the property of the Knights Deep, from which an average of about 1,000,000 gallons of water per diem were hoisted and pumped for some considerable time prior to the war—has been, that the mines, generally speaking, in the deeper sections of ground, make but little water during sinking, so that the question of pumping in the early stages of a deep-level company is not of great importance. In the later stages, however, when active development begins, and many more water channels are cut by reason of extension of workings, the question of the best method to adopt for the raising of water to the surface most cheaply and efficiently, becomes a very important one.

The general practice in all the recent deep-level shafts, is to instal electrically-driven three-throw pumps at intervals of from

500 to 600 feet in the shaft, each pump or set of pumps throwing the water to the one above.

The pumps are placed in chambers adjacent to the shaft, and vary in dimensions according to the quantity of water which it is estimated will require to be handled. In an average case, it is not probable that the water to be pumped—assuming that there are two shafts—will exceed 5000 gallons per hour through each shaft, and, as the pump sumps should not be of less than five hours' holding capacity, they will require to be of sufficient dimensions to hold 25,000 gallons. Making allowance for head-room in the pump chamber, the excavation work would amount in each instance to about 170 cubic yards. In the case of a company pumping only through one shaft, the chambers would of necessity be double that size. The average cost of cutting out the chamber would not be less than 65*s.* per cubic yard, totalling an amount of say £550 in each case. A company pumping through two shafts, each 6000 feet in depth, would require to make provision for 24 pump chambers, each costing £550, equal to £13,200, for preparation of chambers only.

Electric pumps, owing to their liability to stoppage from many causes, should be duplicated at each station, and for spares, there should be provided, say one complete pump for every six installed. The installation for two 6000-feet shafts, therefore, would amount to the following:—

Duplicate pumps in stations	48
Spare pumps	8
Total . .	56 pumps.

The cost of a pump of ample size to handle the amount of water figured on, would be, complete with motor, and fixed in position, approximately £800.

The cost of the cable connecting motors to generators, would not be less than £7500.

The cost of the switchboards, complete with instruments at each pump station, would be about £50, or £1200 in all.

The electric horse-power necessary for running one set of pumps in each shaft, making ample allowances for loss, is 1000, and the cost of a direct connected engine and generator of this capacity would be approximately £12,000. In this instance, as in that of the pumps, provision would have to be made for quick substitution of parts in case of breakdown, or what would be better, a duplicate generator would be laid down.

The boiler plant necessary to furnish steam for 1000 h.p. would cost, erected and complete with economizer, chimneys, etc., £11,500.

Summarized, the total cost would be as under :—

	£	s.	d.
24 pump chambers and sumps	13,200	0	0
56 three-throw pumps, complete with motor, etc.	44,800	0	0
25,000 feet of armoured cable	7,500	0	0
24 switchboards, complete with instruments	1,200	0	0
Two direct connected 1000-h.p. generators	24,000	0	0
Five 200-h.p. boilers erected	11,500	0	0
13,000 feet 6-inch pump column, fixed	5,000	0	0
	£107,200	0	0

The cost of running and maintenance of this plant, including depreciation, could not be reckoned at less than £35 per horsepower per annum, and the cost to a company running 200 stamps and treating 336,000 tons of ore per annum would be at the rate of 2*s*. per ton milled.

The alternatives to the scheme of pumping just described are :—

(1) Installation of high-lift pumping engines.

(2) Installation of Cornish pumping plant.

(3) Installation of (as an auxiliary method) hoisting water by means of tanks affixed to the under side of cages.

High-lift Pumping Engines.

The introduction of mechanically operated valves on the water ends of these engines has resulted in increasing the practical limit of lift to an extraordinary degree.

The pump of the type universally known and used in connection with high-lift work from mines, is the Riedler, first adopted in the mines of Bohemia and Silesia. The chief features of this pump lie in the shape and operation of the valves. These latter are circular in design, and have a lift of from 1 to 2 inches, and are so constructed that they work equally well under all pressures. The valve clearance is large, and in consequence the water passing through is not throttled, tending to efficiency and reduction of friction. The valve opens automatically, but, under the control of a mechanical device, remains open for nearly the whole length of stroke of the plunger, and closes positively at the proper moment by the mechanical controller; the water pounding inseparable from ordinary valves, and the consequent frequent changing are thus avoided. Only one valve is required for suction, and one for discharge in this pump, making the water end comparatively simple. It can be run at high rates of piston speed without subjecting the parts to the severe strains and shocks which are inseparable from the working of ordinary valves, and because of the nature of the valve there is not even at high speeds a calculable slip.

In its usual form this pump is operated by a high-class engine, worked either by steam or reheated air, and is made to work in single lifts against heads up to 3000 feet.

The question as to whether reheated air or electrical power is better, when applied to pumps situated in a mine, is a highly controversial one. The writer leans to the opinion that, in general, electrical power is the more readily adapted to the conditions presented, and would therefore prefer to specify that form of power.

It is clear that the installation of pumps which will raise water from a depth of 6000 feet in two lifts will entail far less expenditure in pump chambers and sumps than will the low-lift pumps previously mentioned. The concentration of the work also in each shaft within two chambers instead of twelve materially reduces the cost of maintenance and supervision, whilst the necessity for plant duplication, inasmuch as it takes effect upon considerably fewer units, would not involve the same large capital expenditure as is required in the case of the low-lift pumps.

Assuming that the first cost of an equipment of electrically-driven pumps with mechanically operated valves were equal to that of an installation of low-lift pumps—which it would not be—the cost of running it would be so low in comparison that, on the tonnage milled by a company running 200 stamps, the charge per ton milled would probably not exceed 1*s*. per ton.

Cornish pumps, which it is not necessary to describe, are essentially low-lift pumps, and expensive to instal on that account. Notwithstanding their great reliability when handling moderate quantities of water, it is improbable that they will be used in the deep-level mines of the future.

The auxiliary method of hoisting water by tanks affixed to the under side of cages is one that is likely to be adopted in every deep-level shaft, inasmuch as, seeing that no additional plant has to be provided, it is at once an expeditious and an economical means of dealing with mine water. Considering that every effort will be made, however, to raise a maximum load of ore, the quantity of water hoisted will probably be kept within the limits of 200 gallons for each hoist. On the assumption that a round trip would be made every four minutes, the double balers would raise 6000 gallons per hour at comparatively low cost.

CHAPTER X.

FINANCIAL PROBLEMS.

The financial problems which face the engineer in the consideration of deep-level property, and which lie no less within his province than the technicalities connected with the engineering branch of the undertaking, are of quite a complex character. The comparatively enormous sums of capital embarked in each venture make it necessary to study exhaustively the following issues:—

(1) The size of property—that is, the area of ground which is necessary—and the potential profit, to warrant the heavy expenditure involved in this class of investment.

(2) The estimated ultimate profit which the property will yield, and how best to arrange the equipment and plan the development, that the said estimated profit may be soonest realized, consistently with a fair redemption of the capital outlay in plant.

(3) The necessity of reaching the producing stage at the earliest possible moment, to obviate loss in accumulated interest and loss by increased charge on tonnage development.

(4) In the consideration of the ultimate profit to be realized, whether the total capital should be redeemed in annual charges, or whether the total profits realized should be at once divided. This point must be definitely settled, before it is possible to calculate on the "present value" of a share which will earn an estimated dividend over a period of years.

It will be useful to illustrate the bearing of these problems on the cases at issue.

In any class of profitable investment, the total profit must be proportionate to the capital sum embarked. In the case of a deep-level company the proportions may be figured out upon the following basis:—

On the debtor side:

(*a*) Capital outlay per claim involved in purchase.
(*b*) Capital outlay per claim involved in equipment.
(*c*) The accumulated interest on the capital invested.

On the credit side:

(*a*) Profit per claim necessary to yield say 7 per cent. interest.
(*b*) Total profit necessary to recommend the undertaking.

Now, it is patent that there is a minimum size of property which can be profitably operated as a deep-level mine. This minimum is found in that area of ore which will return on treatment dividends equal to the sum involved, *plus* 7 per cent. interest over the period that the capital sum has been employed. An average property of 20 claims, for instance, on the third row of deep-levels, would not be expected to yield more than 20,000 milling tons of ore per claim. Assume that the treatment of the total contained ore—400,000 tons—resulted in a profit of £1 per ton, or £400,000. This amount would not return the principal sum, and accumulated interest, involved in the concern, and therefore the operations would result in heavy loss.

Let us assume a block of 42 claims, yielding 20,000 tons per claim, and giving with 200 stamps a treatment of 28,000 tons per month and a profit of £1 per ton, and assuming also a 4000-feet vertical shaft: let us suppose £600,000 to be involved in the undertaking, and a five years' period from the time of commencement of operations until the profit-earning stage be reached, the principal amount of £600,000 being calculated to yield during this period 3 per cent. compound interest—and calculating the total sum accrued at compound interest—to yield 7 per cent. after the beginning of milling operations, and figuring on the interest on the dividends of £28,000 per month at only 3 per cent.—the said block of claims would only just pay back the principal and accrued interest, as shown in example A.

Examples B and C, which follow, illustrate the result of reducing the number of stamps in the same proposal.

EXAMPLE A.

Size of property	42 claims, each 150 × 400 Cape feet or 154·95 × 413·2 English feet.
Tonnage per claim	20,000 tons
Total tonnage	840,000 tons
Number of stamps	200
Tons crushed per month	28,000
Number of shafts	One
Depth of shaft	4000 feet vertical
Cost of shaft (sinking only)	£86,000

Cost of equipment, development, and purchase of property valuing the claims at £5000 each	£600,000.
Time occupied in equipment, development up to commencement of milling operations . .	5 years
Interest on capital sum	3 per cent. compound interest up to time of milling, and 7 per cent. from then to termination of life of mine
Interest on dividends	3 per cent. compound interest
Profits per month	£28,000
Dividends	paid half yearly

Then the capital sum involved up to the time of crushing is £600,000 at 3 per cent. compound interest = £695,564 :

The total tonnage is 840,000, and this is exhausted at the rate of 28,000 tons per month; the life of the mine is, therefore, $\frac{840,000}{28,000}$ = 30 months = 2½ years.

The total capital sum involved at the end of the life of the mine is, therefore, £695,564 at 7 per cent. compound interest for 2½ years = £824,000.

At the expiry of the first period of six months after crushing a dividend amounting to £168,000 would be paid, and so on at the end of each similar period up to the termination of the life of the mine.

To establish an assessment of the value of these dividends on a comparative footing to the sum of £823,600 above given, we must, therefore, calculate the value of the dividends paid *plus* interest at the conclusion of the company's operations:

					£
£168,000	for	2	years at 3	per cent. =	178,231
"		1½	"	" =	175,635
"		1	"	" =	173,040
"		½	"	" =	170,520
"		"	"	" =	168,000
					£865,426

Thus we have £824,000 capital expenditure against £865,426 repaid in profits, which is in effect just a return of the capital sum involved with accrued interest.

EXAMPLE B.

Take the same case on a different basis of treatment, making a difference in the cost of equipment by reducing stamp capacity from 200 to 50.

The cost of the shaft would still remain the same, and this, together with equipment and development and purchase of property, would not be completed for less than £486,000, and the time occupied in equipment and development four years.

The tonnage crushed per month would be 7000, and the profit per ton say 18*s*.

The life of the mine would, in this case, be increased to 10 years.

Taking as before the interest on initial capital required at 3 per cent. up to time of milling, increasing same to 7 per cent. from that time forward, the position is as follows:—

486,000 at 3 per cent. compound interest for 4 years = £546,997.

546,997 at 7 per cent compound interest for 10 years = £1,076,052.

Total value of capital expenditure at end of life of mine = £1,076,052.

The dividends for 10 years at 18*s*. per ton = £6,300 per month or £75,600 per annum on a treatment of 84,000 tons.

£75,600 per annum for 10 years at 3 per cent. compound interest = £866,400.

Thus we have £866,400 total dividends with accumulated interest, as against £1,076,052, capital expenditure, with accumulated interest.

It is, therefore, obvious that the reduction from 200 to 50 stamps does not bring about the desired result.

Example C.

Let us now assume the same case on a 10-stamp basis. It is obviously absurd, but it is interesting to note to what degree.

The cost of the shaft would still remain practically the same, and this together with equipment and development and purchase of the property could not be completed for less than £400,000, and the time occupied in equipment and development at four years.

The life of the mine would again be increased; in this case to 50 years.

The tonnage crushed per month would be 1400, and the actual working profit at a most liberal estimate not more than 10*s*. per ton.

Employing the same factors of interest as before, we have—

£400,000 at 3 per cent. for 4 years' compound interest = £450,000.

£450,000 at 7 per cent. for 50 years' compound interest = £13,255,650.

Total value of capital expenditure at end of life of mine = £13,255,650.

The dividends for 50 years amounting to £8400 per annum on the basis of a treatment of 1400 tons per month would at the end of that time bearing 3 per cent. compound interest be equal to £947,492.

We thus have the value of the capital outlay at the end of the life of the mine, taken at 7 per cent. compound interest, amounting to £13,255,650, as against the value of the annual profits with accumulated interest for the same period £947,493, or a loss of £12,308,157.

In each of the three examples given, 7 per cent. interest has been charged on the capital, as that is the minimum amount looked for in a mining proposition, whilst 3 per cent. only is allowed on the dividends, that being nominally bank rate.

The profit per ton allowed in each case is only at most an approximate figure, but the margin is well on the safe side. In example C, for instance, it is doubtful whether on actual milling the profit per ton would be anything like 10*s*., seeing that the costs would be so much greater in proportion to the 200 stamp basis.

From these three examples it is clear that there is nothing to be gained by erecting a smaller milling plant than that specified in example A, and although of the three examples this is the most feasible, it is equally clear that the expenditure necessary is too heavy for the calculated profit. The time taken to work the block out on the basis stated would be only two-and-a-half years from the time crushing began, and therefore the writing off of the plant at the rate of approximately 45 per cent. per annum would be most unbusiness-like in method. In order to get full value out of the plant it should run for 7 to 10 years, or three to four times as long as the 42 claims would allow of; and therefore, as a business proposition, the size of the property on this stamping basis should not be less than four times 42, or 168 claims, which would give the company a life of 10 years.

The following examples—D and E—serve to compare the difference between a 40 and a 100 claim deep-level proposition.

EXAMPLE D.

This is a case of a property 5000 feet from outcrop, in which an average dip as from outcrop is maintained at 40 degrees. To intersect the reef in this property it would be necessary to sink a vertical depth of 4195 feet. Such a shaft would cost approximately £85,000, exclusive of shaft equipment, the latter factor varying slightly as the number of stamps to be run. Supposing the size of the property to be 40 claims, estimated to contain 20,000 tons per claim. Sixty stamps would be the most such a property would reasonably support, and the life of the mine would be about eight years.

Inclusive of shaft sinking and equipment, the total cost to reach the development stage would be about £180,000.

This estimate is made up as follows :—

	£	s.	d.
One shaft, 4195 feet, at £20	83,800	0	0
One permanent headgear	6,500	0	0
One temporary headgear	1,000	0	0
One temporary hoist, with boiler and building . .	3,000	0	0
Pumping plant	50,000	0	0
Workshops, stores, and tools	5,500	0	0
Buildings	5,500	0	0
Permanent hoisting engine	7,500	0	0
Permanent auxiliary hoisting engine	4,000	0	0
Boilers, economizers, chimneys, etc.	6,500	0	0
Small air compressor, boilers, piping, etc. . . .	4,000	0	0
Sundry plant, rope, trucks, rails, sinking pumps, piping, etc.	3,000	0	0
Portion of permanent compound	1,000	0	0
	£181,300	0	0

Say £180,000.

All this expenditure is incurred in arriving at a stage where development could be started, and the cost of development, reducing and treatment plant, etc., would begin from that point. The property is, therefore, committed to an initial expenditure of £180,000; and the subsequent expenditure in equipment, development, dams and reservoirs, electric plant, etc., on a 60-stamp mill basis, would figure at about the same amount, or a grand total of £360,000.

We then have to consider the price of the claims, and this may be placed at say £3000 per claim, or £120,000 for the 40 claims.

The total expenditure, therefore, necessary to purchase the ground, and equip and develop same up to the time when milling should commence, would amount to £480,000, exclusive of interest on capital up to the time named.

On 40 claims the capitalization per claim amounts, therefore, to £11,500 per claim, and the nominal capital of a company to work it on this basis could not be less than £500,000.

We have assumed the tonnage per claim to be 20,000, and we shall further grant that the profit per ton may be 15*s.*

A 60-stamp mill would crush 8500 tons per month, or say 100,000 tons per annum, equal to a profit of £75,000 per annum, or 15 per cent. on a capital of £500,000. (The average recovery for 11 deep-level properties in 1898 was 48*s.* 6*d.*)

The dividends regarded as an annuity over the life of the mine would have a present value of £435,060, making allowance for amortization fund at 3 per cent., and taking the dividends as 6 per cent. the annual payment to amortization fund would be £48,900. This amount, invested annually at 3 per cent. compound interest, would amount to the present value at the end of the life of the mine.

In view of the fact that the present value of the dividends for the period named is less than the nominal capital of the company, it is obvious that the size of the property does not justify the expenditure of the amount of capital set against it.

EXAMPLE E.

If we now assume the property to be one of 100 claims, the difference in favour of the larger property becomes at once apparent, simply because we have a larger area over which to distribute the initial capital expenditure.

Calculated on the same basis as before, the proposition is as follows :—

Size of property	100 claims
Distance from outcrop	5000 feet
Average dip	40 degrees
Vertical depth to reef	4195 feet
Cost of shaft (including equipment)	£200,000
Tonnage per claim	20,000
*Profit per ton	15*s.*
Number of stamps	150
Life of mine	8 years

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* As costs would be lower on the larger number of stamps, the profits would be proportionately higher. No allowance is made for this.

Expenditure up to time of milling	£500,000
Price of claims, each	£3,000
Total expenditure	£800,000
Capitalization per claim	£8,000
Nominal capital	£850,000
Profit per annum	£187,500
Percentage profit	22 per cent.
Present value of annual dividends, making same allowance for Amortization Fund as before	£1,087,650
Annual payment to Amortization	£122,351

It will be observed that in this case we have a present value of £1,087,650, on a nominal capital of £850,000, after allowing for £160,000 capital expenditure in excess of the former case. It will be noticed further, that the capital expenditure in this case, exclusive of the price of the claims, amounts to £5000 per stamp on a 100-stamp basis.

This figure is approximately the average to be deducted from the statement of expenditure on deep-level properties on the Witwatersrand at the present date.

In the case of the 40 claims and 60 stamps, the equipment and development figures at £5800 per stamp.

It has been shown that a company of 40 claims containing 20,000 tons per claim valued at 15*s.* per ton profit, situated 5000 feet from the outcrop, where the average dip is 40 degrees, is not a paying one; whilst, on the contrary, if the size of the property is increased to 100 claims it becomes a sound paying one.

The factors of value assumed are based on the data available at present time, and regarded from a general standpoint, may be taken as in the neighbourhood of average. On this basis, therefore, it is evident that we require at least 100 claims to make the scheme a profitable one. As between mine and mine, however, in individual cases, there will be found an infinite divergence in all of these factors, and each case must be treated on its peculiar merits. That is to say, it is impossible to establish any rule to apply equally to all properties situated at a distance of 5000 feet, or any other distance, from outcrop. For instance, in the two cases exemplified, the former may be in a neighbourhood where the grade is such that a profit of 30*s.* per ton may be looked for, or the reef thickness may prove to be equal to 40,000 tons per claim. Either of these variation make all the difference to the proposition, and 40 claims containing 40,000 tons per claim, capable of yielding a total recovery of 60*s*. per ton, is obviously a much more valuable proposition than 80 claims at 20,000 tons per claim, at the same recovery value per ton.

In regard to capital expenditure, it is just as obvious that the potentialities of the proposal treated of in these remarks do not admit of or justify the outlay of £1,000,000 working capital, any more than that of only £50,000; and following on this statement, it is equally clear that there must be a mean basis between these two extremes, calculated to yield the maximum profit.

The problem, stated briefly, is that, since there is an initial expenditure of something like £200,000 to be incurred for each shaft, in cases where the reef is to be intersected at a depth of something like 4000 feet, the size of the property, having due regard to the factors of probable value and thickness of reef, must be such as will permit of a certain reasonable life, on a basis calculated to reimburse the whole of the capital embarked with interest.

Seeing, therefore, that it is impossible to lay down any rule in the calculation of the capital expenditure warranted in a given mining proposition, we must treat each case on its individual merits, and in that light estimate—

(1) The ultimate net profit that the mine will yield, based on the calculated probable profits per ton of ore contained.

(2) The annual profit that such an amount would represent on a life of 10 years.

(3) The present value of such profit regarded as an annuity over the period named, allowing for amortization of capital.

(4) The equipment and development required to work out the mine in a stated period.

(5) The cost of such equipment and development up to the time of starting actual milling operations, inclusive of compound interest.

(6) The capital of the company, covering the cost of equipment and development, and the purchase price of the property.

Having these data in hand, a comparison of the present value of the annual profit—after allowing for amortization of the capital—as against the capital required, will reflect the value of the shares, and the general value of the proposition.

Naturally the value of the calculation depends entirely upon the factors of tonnage and profit per ton.

The period of 10 years is taken to cover the depreciation of plant, and this may be varied as is warranted by the instance.

Needless to say, all estimates must of necessity be only accurate in so far as the data upon which they are built prove to be correct, and the innumerable factors of divergence encountered in all mining work preclude the possibility of making anything but approximate estimates of the value of unexploited ground.

In framing the above basis of calculation, therefore, all that is claimed for it is that it establishes a ground-work upon which one may, with some show of reason, fix the limit of capital to be embarked, and decide upon the number of stamps to be erected. As matters are to-day, the decision as to these important points is frequently more influenced by sentimental than by material reasons.

In considering the estimated profit which a property will realize, the engineer is aware that he is dealing with some unknown factors, the influence of which upon his calculations it is not always possible to foresee. These are primarily :—

(1) Variations from the estimated average in the ore grade, and in the reef widths.

(2) The encountering of heavy faulting, or dyke intrusions, resulting in the alteration of the whole plan of development.

In planning the development of the mine, and in estimating upon the equipment required, the engineer therefore makes large allowances for unforeseen difficulties which may have the effect of reducing the average grade of the ore, and the average payable tonnage per claim. But when he has arrived at a safety factor under this head, there is, on the other side of the balance, a credit factor which he generally leaves entirely out of calculation. I mean the advent of conditions which will result in transferring very large quantities of ore, at present written down to the unpayable total, to the payable section, as also steepening of dip and increased width and value in reef, etc.

As at present conditioned, it is the writer's practice to write down all ore under 8 dwts. assay value on stoping width as unpayable, and every month a certain tonnage of ore which averages, say 7 dwts. assay, is thus dealt with: aggregating in a term of years, a total which may run into very large figures. In estimating the total tonnage to be treated, and therefore upon the total profit which it is expected a property will yield, the possibilities which attach to the profitable treatment of 7 dwts. rock are not figured in; and consequently, should the succeeding years witness that general reduction in working costs, which there is every reason to anticipate, the ultimate profit realized would be the more by that amount of ore below 8 dwts. in value, which will be dealt with in the reduction and treatment plants.

Having arrived at an estimate of the ultimate profit which the mine will yield, it is next necessary to consider in what manner that ultimate profit may soonest be realized—that is: Supposing the estimated figure to be £20,000,000, how much capital expenditure should be carried out to earn that amount in the earliest period?

It has been shown in a former paragraph that the minimum recommendable size of deep-level property on average grade and tonnage per claim is one in the neighbourhood of 100 claims, which on a 150-stamp milling basis possesses a life of 8 years, whilst a reason is given for so arranging equipment as to make the life 10 years. The custom, however, is rather to lay out properties with longer estimated "lives" than 10 years; 20 years being nearer the average. The advantage of the longer period —assuming always that only 150 stamps are run—lies in the increased possibility of the company sharing in the economic reforms which such a period is bound to witness, and also in the lighter claim capitalization under the head of equipment-cost distribution. The question arises, however, at this stage, as to the relationship of a 200-stamp mill equipment to a property which will sustain it for 20 years, or will yield 6,720,000 tons of payable ore; and whether any better basis of equipment can be provided.

There are three principal factors entering into the question of mill supply, which are:—

(1) The average width of the reefs.

(2) The possible rate of development.

(3) The percentage of payable ore in a given developed area.

It is evident that a reef which will allow of stopes 6 feet in width will supply the mill requirements from fewer points than one which can only be stoped 3 feet in width; the latter in turn would supply ore at a quicker rate than a stope on a thin reef, say 12 inches in width, worked on the "resueing" system—that is, a system which involves mining the foot-wall rock of the reef separately from the ore, and disposing of the rock thus mined in filling in the stopes, subsequently blasting out the reef matter as a clean body.

The figures of tonnage per claim on each reef at an angle of 20° would be, approximately, as under:—

Reef 6 feet in thickness	33,000 tons per claim.
Reef 3 feet in thickness	16,500 " "
Reef 1 foot in thickness	5,500 " "

It is patent that in the last-named instance it would be impracticable to supply a 200-stamp mill, since the rate of development necessary would be one claim in every 4 days, and therefore, be the property of sufficient area to furnish the total tonnage before stated of 6,720,000 tons, the ore which it would furnish per month would probably not supply more than 50 to 60 stamps.

The possible rate of development depends, however, as much upon the system of attack as upon the thickness of the reef; or, stated as a proposition, it may be said that the number of points of attack required in any reef varies as the width of ore and the mill requirements. It would be possible, doubtless, by placing levels and winze connections at short intervals, to keep pace with heavy mill requirements, but the scheme would not be one consistent with well-planned working, and the cost of development would be excessive. The only way in which a quick rate of development can be secured, having in view all contingencies, is to push the shaft sinking forward at such a speed that the intervals for successive levels are quickly gained, and, to accelerate the opening up of the different levels, winzes may be sunk at selected points, equipped with hoisting apparatus: these giving, when they have reached the proper depth to connect to the work from the shaft, two more working faces. It is all-important, therefore, that the shaft should be kept well below the stoping place, if any great rate of development in advance of a fixed mill supply is to be secured.

There still remains, however, the question of what percentage of the ore developed by a given amount of driving is payable. This is a point which every mine has to determine by actual disclosure of the values, and it depends on the width of the pay-shoots as compared to the total width of the property. A mine, for instance, from which is only obtained 30 per cent. of payable ore out of the total developed, would require, firstly to return exceptional values to warrant its development, and secondly, would probably not be able to supply a big milling installation.

The foregoing paragraphs make the fact sufficiently clear, that for every mine there is a certain development which can be achieved by systematic procedure, and this must, in every instance, be carefully studied when contemplating mill erection.

The questions above considered have a very important bearing upon the problem of how best to plan the development and equipment, that the ultimate profit which the mine is expected to yield may be soonest realized, consistently with a fair redemption of the capital outlay on plant.

In considering this question, let us assume that we are dealing with a deep-level mine in the Central Witwatersrand which has an area sufficient to produce payable milling ore for 200 stamps for 20 years. Let it be assumed that the vertical shafts are sunk, and development of the mine proper is about to begin, and the value of the ore, as figured from the results of the producing mines in the immediate vicinity, has been arrived at. The first question that arises is, How soon may milling commence? This would firstly be dependent upon the conditions which were disclosed at the points where the reef was struck—that is, as to whether or not the country had been subjected to faulting; and secondly, as to whether the ore grade indicated a confirmation of the estimates which were fixed, based upon the results obtained from the neighbouring mines.

If all the evidence pointed to the existence of normal conditions, then milling plant would be immediately ordered, and at 12 months from the date of order, given favourable conditions —all plans in readiness and materials awaiting shipment —it would be possible to start milling with 100 stamps, which would be gradually increased to 200 stamps as the development progressed. Let it be assumed that at the time the 200 stamps begin running the mine has a life of 20 years, and let the profit, on a treatment of 28,000 tons per month, be £28,000, equal to £336,000 per annum. At the end of 20 years the total profit earned by the company would be £6,720,000. Now, if it were possible so to increase the rate of development, and, *pari passu*, the milling capacity, that the total profit could be earned in half the time, the nett difference in the company's favour would be that amount represented by the compound interest on the capital invested, *plus* the interest on the annual dividends, over the final 10 years, whilst against this must be set the extra capital necessary for increased equipment and development, plus interest on same.

The advantage the shareholders would gain might be thus stated. Supposing it were possible to discount the total earnings of the company as at the end of 20 years. The present value of the amount of £336,000 for 20 years, calculated to yield 6 per cent. per annum, and allowing 3 per cent. for amortization, is £3,456,768. If, however, the total amount of £6,720,000 were discounted as a yearly dividend of £672,000 for 10 years at the same rate of interest, the present value of the amount would be £4,564,224, or £1,107,456 greater than the same amount earned in a 20 years' period. The additional capital expenditure necessary to do this would amount to say £400,000. The inducement to realize the ultimate profit in the shorter period is obviously a very great one, and the possibilities of achieving it must receive the greatest attention.

It has been shown in former paragraphs that for two reasons —namely, the saving of interest and the lightening of development cost—it is urgent that a mine should enter the producing stage as soon as there is a regular average ore supply available. In the writer's opinion the best solution of the problem would be the following :—

In the first instance we assume that the first year's development has put 200,000 tons of ore in sight. In the meantime the first 100 stamps have been erected, and the plant is so arranged that additions to it at six-monthly periods are made —that is to say, during each six-monthly period there is unceasing plant erection, and the coupling up of additional plant takes place twice a year.

The mine development facilities would improve from month to month, as new points of attack were opened up from the shafts, and each succeeding half-yearly period would witness a growing tonnage development. As the development progressed, so in strict proportion would the milling capacity be increased, until a time was reached when it was estimated that the mine had only seven years' longer life. It would then be necessary to cease all further expenditure on increase of milling capacity, in order that the capital so expended might be, to a great extent, written off during the remaining years of the company's existence. By adopting a scheme on these lines, at the end of five years the company would have erected 450 stamps, and would have crushed at the end of the said period 2,010,000 tons of ore, leaving a balance of 4,710,000 to be dealt with. At the then rate of crushing, which would total 756,000 tons of ore per annum, the life of the mine would only be 6·2 years—which is rather under the 7 to 10 years we require for redemption of the capital outlay on plant—or, the total tonnage, and consequently the ultimate profit, would be secured in 11·2 years, as against 20 years working on a fixed 200-stamp basis.

Notwithstanding that the writer does not lose sight of the great undertaking involved in the feeding of a 450-stamp equipment, nor of the heavy capital outlay necessary to accomplish it, it is his firm conviction that deep-level mining on the Rand will be prosecuted on quite as extensive a scale as that he has just pictured, though in every instance it will be necessary, as has been shown, to closely study the general conditions in the mine, so that the equipment provided will correspond to its possibilities of practicable supply.

Expenditure on Capital Account.

The custom of placing certain amounts of monthly expenditure to "capital account," after a mine has secured a reasonable development, and has proportioned its equipment accordingly, is one which is open to a certain amount of abuse.

The existence of a capital account, on a mine which has finished with erection work, offers great temptations to a manager to charge more than a fair proportion of his monthly expenditure to the said account, in order that his operating expenses may be lower; and for this reason, apart from others which will readily suggest themselves, there will not be that careful control of details of expenditure which are necessary, and which must be exercised by a manager who has to balance his total expenditure against revenue at the end of each month. But there are cases in which, in the opinion of the writer, it is quite necessary to carry a capital account, these being in those instances where there is a heavy gain of development as against tonnage crushed, and a regular increase in the milling equipment. In such cases, were all the expenditure charged against revenue, it would result in absorbing the total distributable amount, to the detriment of the present shareholder.

In cases, however, where mines have reached their full milling equipment, and the development work is so far ahead of the mill that the mine yields can be averaged from month to month, and hence when there is no necessity to develop more than the monthly tonnage crushed, then all capital expenditure should cease, and the total outlay, in whatever direction, should be made a charge on the current revenue.

Experience thus far on the Witwatersrand teaches us, however, that as we progress our requirements increase, and whether it be a new shaft, or some other heavy item of expenditure, there is ever arising a need for a capital sum, which cannot under any pretext be chargeable to revenue; and thus for the reasons already stated, the probabilities are that a capital account, or its equivalent in some form, will always figure in returns of expenditure.

The importance of the question in its relation to profit and loss, and hence, to revenue, is obvious, seeing that these all important accounts may be influenced very largely according to the particular procedure adopted, and it is therefore incumbent upon those responsible for the administration of a mine, to observe the greatest discrimination in the employment of the charge.

In principle, and as a general statement, it may be said that outlay on capital account can be applied only to items which represent an added value to the property extending over its life.

It is not always an easy matter to classify expenditure, and occasionally instances arise where the correct allocation of each item becomes a most difficult problem.

The administrators of a company are naturally anxious to avoid undue taxation of the shareholders, and the tendency is, therefore, to favour the revenue account by charging certain classes of expenditure to capital, especially in those cases where the margin of profit earned is very small. Whilst condemning that tendency, it must be allowed that in cases where the pros and cons after careful deliberation appear to be evenly balanced, criticism should not be too harsh. Strictly speaking, however, nothing should be made a charge on capital account which does not rightly belong to it.

All other expenditures, such as maintenance, renewals, etc., come under the head of revenue expenditure.

We have three methods illustrated in practice to-day, in which expenditure on capital account is treated differently.

These are briefly :—

(1) All outlay up to the time of milling operations charged to capital account; all subsequent expenditure charged to revenue.

(2) All outlay up to the time of milling operations charged to capital account, with provision in the shape of a special fund for all subsequent expenditure not rightly chargeable to revenue.

(3) A continuous capital account to which all expenditure outside of revenue is chargeable.

In the first case the scheme is perfectly fair in the absence of any extraordinary expenditure, but supposing it is considered advisable to increase the milling capacity or sink a new shaft, the company would either have to absorb all its revenue, until the extra outlay was met, or it would be forced to make some financial arrangement for the distribution of the charge. The former course would be unfair to shareholders, and the latter would be a violation of the principle.

In the second case, it would be difficult to frame a basis for the establishment of a fund to meet either unforeseen contingencies or those which, though being foreseen, cannot in the nature of things be gauged. A fund may be established and never utilized, or it may prove to be totally inadequate to the company's requirements.

In the third case, there is always the danger of abuse in the shape of the charge of revenue expenditure to capital account.

Naturally, any principle is open to abuse, but the more sound and practical the principle the less the danger.

In the opinion of the writer it is impossible in this, as in other matters influenced by varying factors, to define any rigid principle to be adhered to, but in the main it would appear that the most commendable policy is—

(1) To charge all outlay up to the time of milling to capital account.

(2) All subsequent working costs, including maintenance and depreciation, to be defrayed by revenue.

(3) All new extraordinary outlay to be provided for by the creation and disposal of reserve shares.

It is assumed that no extraordinary expenditure will be incurred that will not pay for itself. The shareholders will, therefore, not be in any way injured by this procedure, whilst, on the other hand, if the new outlay is really warranted, they should benefit notwithstanding the increase in capital.

The more that our mining propositions on the Rand are regarded as industrial enterprises of limited terms of existence, the sounder will the administration of their financial affairs become, and the confounding of book profit with net profit, which is an inseparable feature of some systems of accounts now in vogue, will make place for a concrete statement of the company's affairs and the results of its operations, which may be read at a glance.

One aspect of the misuse of capital expenditure is instanced in the following example :—

A company equips and develops a property, say on a 100-stamp basis, and in so doing spends £400,000 up to the time when milling operations begin. Assume that the whole of this amount is charged to capital account and all subsequent expenditure charged to working costs, and that development amounting to 300,000 tons has been completed.

Crushing begins, and 15,000 tons of ore per month are taken to the mill for treatment. Profits of, say £18,000 per month are made, and dividends are declared at regular intervals. If the mine is being worked properly, the development is proceeded with at the rate of not less than 15,000 tons of milling ore per month, and the machinery and plant and general equipment are kept in thorough repair, the cost of such development and maintenance being deducted from the monthly revenue. Operated on this plan the dividends available are a true profit.

It is not difficult to conceive, however, that this working policy is occasionally neglected : that is to say, instead of keeping up the development to 15,000 tons per month, it is either partly or wholly suspended, whilst only such maintenance and repair work is carried out in connection with the plant and equipment as will suffice to keep it running. The result is that large profits are made and handsome dividends declared for a time, perhaps a few years, then the company finds itself at a standstill for development and equipment, with reconstruction staring it in the face, and this, notwithstanding the fact that the property may be a large one with payable ore showing in the majority of the faces of its undeveloped sections.

Briefly stated, the result of the operations is that, of the £400,000 originally expended, say £200,000 has been paid back in the shape of dividends, but the company is so placed that a further expenditure of £100,000 on development and overhaul is necessary to enable it to continue profitable work.

This disappointing condition of things is directly traceable to undue haste in the desire to make large profits from the moment milling began. With 300,000 tons of payable ore developed, the company's administrators have not considered it either necessary or politic to continue development, and hence the standing development is drawn upon to meet the milling requirements. The cost of development having already been met in capital account, the ore going to the mill will only bear a comparatively low charge. Doubtless, too, the higher grade of ore would be mined to compensate for leakages inseparable from a new plant.

The profits under these conditions would naturally be high, but the time would inevitably come when the administration must face the question of development and repairs. It would be realized that unless these two vital factors were attended to, a point would sooner or later be arrived at, when there would be no alternative but to shut down for a period. On the other hand, the administration would be fully alive to the fact that consideration of these matters would mean increased monthly expenditure, and a proportionate drop in the profits. The latter contingency would, to those not acquainted with the position, imply a falling off in grade, and would, therefore, damage the company in the eyes of the public. The position would be a difficult one, and one calling for delicate handling. There can be no question, however, as to what should be done. The mistake, once recognized, would have to be remedied by attacking the neglected work without further delay. The result would be a falling off in profit, and the company would have to confess that it had got behind with certain work, to recover which would temporarily absorb most of the revenue.

If the company in its desire to make high returns had not set itself an impossible standard of monthly profits to sustain, the position might have been saved, but after working at the lowest cost on its highest grade ore, it left itself no latitude and therefore no alternative.

It is possible in an exceptional case that the administration would not realize the difficulties to which neglect of the work mentioned would lead them, until the position forced itself upon them. Whatever may be the cause, however, there is no gainsaying the fact that such a position is unsatisfactory, and from the engineer's point of view cannot be excused, whilst in its financial aspect there is much matter for criticism.

Depreciation.

Depreciation may be stated as an amount calculated to represent the periodic diminishing value of machinery and plant or other assets to be regularly charged against the book value of same. The amount naturally varies in direct ratio to the estimated life of any given asset, and it is, therefore, impossible to apply any set general rule to meet the requirements of all the mines in regard to the charge. On the other hand, it would complicate matters endlessly to deal with each item separately on any given mine, and in each case the accepted procedure is, to generalize the charge according to circumstances, and, it may be, under one or more heads.

Elsewhere it has been shown that the life of a mine should be so regulated, in the matter of claim area and equipment, as to ensure from 7 to 10 years, in order that the writing off on the score of depreciation may not be an unduly heavy charge. In other words, the main items of which the equipment consists are estimated to last for the period named if properly maintained. The residue value of such assets may or may not be an appreciable or calculable amount: in the majority of cases it will be neither.

It has been pointed out on more than one occasion that depreciation has never yet been charged on the cost of claim areas, and this despite the fact that a property is being actually worked out at the rate of so much per cent. per annum. For instance, take a property of 90 claims estimated to contain 20,000 milling tons per claim. Working on a 100-stamp basis, 180,000 tons per annum would be crushed, equal to 9 claims; and in 10 years the mine would be exhausted, and yet at the end of the 10 years we find the property figuring in the balance-sheet at the same value as it did before a ton of ore had been extracted. Such methods of accountancy would appear at first sight to be absurd, but a close study of the problem will prove that there is no feasible alternative. If such a charge were made it must be based on an estimate of the life of the mine, and as any such estimate could only be the merest approximation, the charge made must necessarily be of the same nature, and therefore open to the same objections as have been raised in regard to amortization of capital. To-day we estimate the life of a mine on the conditions now obtaining, and for all practical purposes this must, for to-day, suffice; at the same time it is patent that the trend of our endeavours is in the direction of decreasing costs and, *pari passu*, increasing profits, and every step thus gained brings into the range of payable ore, increased tonnage, and therefore prolongation of the life of the mine. There are many other possibilities which vitally affect the question, and for these, and the other reasons given, depreciation upon property cannot, in the writer's opinion, be charged.

It is admitted that a charge for depreciation should be made upon certain assets, and that such charge must be deducted periodically from the book value. The profit and loss account reflects this deduction, and the amount itself should be placed to the credit of a special fund to be utilized in providing such additions and renewals to plant, etc., as may from time to time be required. If a company makes no allowance for depreciation, such additions and renewals must either be paid for on capital account or charged to working costs. In the latter event the spirit of the depreciation charge is fulfilled, but as heavy renewals may be occasionally necessary, it is advisable to distribute the expenditure over a period or to have a special fund upon which to draw, as otherwise the returns of cost and profit per ton will vary unnecessarily.

Maintenance.

The cost of maintaining machinery and plant in working condition is directly chargeable to working costs, and may be treated either as a separate account or be distributed over the various heads upon which it is expended.

Reserve Fund.

It is usual, and is recognized as necessary and legitimate, to establish a fund or account called "Reserve," the functions of which shall be to equalize yields upon certain grounds and within certain limits. Thus a company may one month exceed its average return in gold won by 10 per cent. In lieu of declaring this in its profits, the amount is carried forward to a reserve account. The following month the returns may be 10 per cent. less than average, owing to shortness of labour, or to a few days' stoppage, or to some unforeseen hindrance, but having 10 per cent. in hand from the previous month, it is able to declare an average return, whereas if the full amount won in the previous month had been declared, the return would have dropped considerably and possibly to the detriment of the value of the company's shares.

The fund is not established with any other design than that described, and whilst this principle is adhered to, its use is not open to abuse.

Amortization of Capital.

In any class of financial undertaking which has a limited period for profit-earning, the question is sooner or later raised as to the desirability or otherwise of redeeming the capital concerned by annual charges over the term during which it is estimated that profits will be earned. *Primâ facie* it would appear desirable that an investor should, at the end of a company's career, have something more in which to participate than the realizable assets then possessed, such as plant, water-rights, etc., etc.; and, in fact, it is not infrequently urged by accountants that the total original capital should be available either for redistribution or for reinvestment. There are, however, certain very cogent reasons why amortization should not be recognized in practice, by the company in its corporate sense, although in principle, by the individual shareholder, it should be observed.

It is patent at the outset that the practice of amortizing the capital outlay by annual deductions from the profits serves to diminish the current dividends to a company's shareholders—that is to say, an amount of money is annually retained in order to build up a solid sum which will equal the original amount invested when the earning period of the concern is finished. The sum so retained from shareholders might well be utilized by them, in their individual capacity, in an immediate investment which would, possibly, earn considerably higher profit than that which could be obtained from the reinvestment of the total capital sum, especially in view of the fact that the trustees for such a fund could only invest in the very safest, and, therefore, the lowest interest-paying securities, and in such case the shareholder would be a distinct loser by the retention.

The loss would be greatly accentuated in the case of investors who bought interests in the property in the early stages of its history and later sold out, unless the fact that amortization was being provided for reflected itself appreciably in the value of the shares. All such investors are paying away a share of profits—which rightly belongs to them for their immediate use—for the benefit of those who at a later date buy the company's stock; in fact the arrangement gives a benefit not due to those investors who become interested in the property towards the end of its career.

The shareholder who receives his share of the full current profit may, if he wishes, reinvest a portion of it in funds for the purpose of amortizing his capital outlay, and at least he has the immediate control of his capital's earnings.

But, be the arguments *pro* and *con* what they may, it cannot be denied that a company which frankly confesses that it makes no provision for amortization is just as sound from the investor's point of view as one that carries annual amounts to an amortization account, for the simple reason that in both cases there is a proportionate share value, which can without difficulty be figured out.

For instance, take the case of a company with a capital of £400,000, with a life of ten years, and the profits whereof amount to £200,000 *per annum*.

FIRST CASE: ALLOWING FOR AMORTIZATION.

Take the rate of interest for amortization fund at 3 per cent., and the rate of interest for dividends at 6 per cent.

The present value of the net profits—£2,000,000—to be earned during the life of the mine on this basis is £1,358,400.

The annual payment to amortization fund will amount to £118,452.

£81,548, or the equivalent of 6 per cent. on the present value, would be paid yearly as dividends.

The amortization fund in this case is an amount deducted from the yearly profits, which, if invested annually at a stated rate of interest, will amount to the present value when the mine is exhausted.

On this basis the value of the shares in a capital of £400,000 would be £3 7*s.* 11*d.* per share.

If we take the capital of the company, viz. £400,000, as the amount to be redeemed, whilst the balance of the profit is to be paid away as dividends, then we must deduct from the yearly profits the sum of £34,880, which amount, compounded annually at 3 per cent., will amount to £400,000 in ten years.

The balance of the profit, namely £165,120, equal to 41 per cent., would then be available for distribution as profit.

The shareholder receives a dividend of 41 per cent. per annum on his £1 share, and in addition he will, at the end of the life of the mine, receive £1 for every share.

The total net profits distributed as dividends at the end of ten years will be £1,651,200.

In the one case only £81,548 is distributed, and on the balance 3 per cent. is obtained, as against £165,120 in the other case, with the same interest on the balance.

The present value of the shares in either method would be the same if shareholders invested their profits received at only 3 per cent.—the rate fixed for amortization—but if a man could invest his money at, say, 6 per cent., then the second statement would suit him better than the first.

SECOND CASE: NO ALLOWANCE FOR AMORTIZATION.

In this case the shareholder is to receive the whole of the profits, viz. £200,000 per annum, or 10*s.* per share.

The total value of £200,000 received annually for a period of ten years, compounded annually at 3 per cent., is, at the end of the period named, £2,292,760.

The present value of this amount is £1,706,042. To find the value of the shares allowance must be made for the redemption of the capital. It has been shown that £34,480, set aside annually and compounded at 3 per cent., will amount to £400,000 in ten years; therefore from the total of the present value we must deduct ten contributions of £34,480 to arrive at the real figure of present value. Thus, £1,706,042 less £344,800 equals £1,361,242.

Value per share on a capital of £400,000 equals £3 8*s*. per share as before.

From these two statements it is clear that the shareholder—who, after all, is the person to be considered—need not be misled as to the value of his shares when no allowance is made for amortization; and, on the other hand, he obtains the use of the whole of his share of the profits to reinvest or do with as he wishes.

Those shareholders who insist upon the carrying out of the practice of amortization will discover that there are many almost insurmountable obstacles to be dealt with in defining the amount to be set aside annually, seeing that the factors of "life" and "value" can only be approximately defined. We estimate, for instance, that the "life" will be 10 years, and the distributable profit £200,000, and on that premise we proportion the contribution to the amortization fund. Eventually the "life" may prove to be only 7 years, and the profit £100,000, or it may be 20 years and a profit of £300,000. The fact that such variations may occur is in itself sufficient to demonstrate the impracticability of the adoption of any such proposal. The charge decided upon to-day may subsequently prove to have been unduly high, and the shareholder of to-day who disposes of his holding will have been deprived of profit which has been carried forward for the benefit of a later holder.

The value of the shares when the company is not amortizing its capital would gradually drop towards the end of its life, in strict proportion to the ultimate amount which it was estimated would be available for distribution.

Mine Development Redemption.

In the Witwatersrand mines an item of the cost per ton of production is that of mine development redemption. The principle of charging a fixed sum in working costs for each ton of ore milled from month to month avoids the undesirable method of charging the whole lump sum—which probably in most cases covers a considerable reserve tonnage over and above the monthly tonnage milled—against the current monthly revenue, which would result in unduly taxing present shareholders for the benefit of posterity.

The amount charged to mine development redemption in any given mine varies with the costs of development and the amount of tonnage which a certain quantity of driving, winzing, rising, and cross-cutting blocks out. A mine that has wide shoots of pay ore, and but short intervals between the shoots, will develop a much greater milling tonnage—assuming equal widths—than one which has narrow and widely separated shoots, and hence the cost per ton of ore is reduced.

To illustrate this, let us assume that we have, on the one hand, a mine that has driven two levels of 2000 feet each in length, the one under the other, and that in the stated length there are two shoots of payable ore, each 700 feet in width, separated by 600 feet of unpayable rock. It is assumed that each level has 150 feet of ore "backs," and therefore the case of one will illustrate the instance.

In a mine developing during milling operations the costs may be set down as approximately 70*s.* per foot. The driving of a level 2000 feet in length, with two connecting winzes, each 150 feet, would cost £8050. Assume that the reef averages 30 inches of milling ore—allowing for all deductions of sorting, pillars, faults, etc.—in the pay shoots. Then $1400 \times 2\frac{1}{2} \times 150 = 525{,}000$ cubic feet, which at 12 cubic feet per ton equals 43,750 tons. The total cost to develop 43,750 tons is £8050, or at the rate of 3·7 shillings per ton. This figure would therefore be that at which the development of this tonnage should be redeemed in the working costs. In the second case, let us assume that the mine has two payable shoots, being 400 feet each in width, and separated by 800 feet of poor rock.

The cost of development would be the same as that in the first instance, namely, £8050, and on the same assumption of milling width the drive would develop 300,000 cubic feet of ore, which at 12 cubic feet per ton is equal to 25,000 tons of ore, and this divided into the total cost averages 6·4 shillings per ton, which is the amount that would be charged against this block for mine development redemption.

It is usual in figuring the costs of development, however, to take the total tons developed over a given period as against the total cost incurred, and thus establish a figure, for inclusion as a fixed charge in the working costs. As a matter of fact, the writer is very doubtful if the amount spent in development is redeemed by the charge made for mine development redemption excepting in rare instances. The usual manner of charging the amount is as against the total tons along the stretches driven, making no allowance for sorting, unpayable ore, or pillars, and the consequence is that a great reserve of ore is amassed, and appears on the company's books ostensibly as a valuable asset, which sooner or later has to be written off as valueless.

There is always, however, the possibility that a certain proportion of ore which is left in the mines to-day will one day be profitably treated, and in that case, having no development redemption charge to pay (as this is presumably paid on the tonnage already treated), and in many cases being less expensive to mine (as it occurs in stopes from which the more valuable ore has been taken), the potential profit is considerably augmented.

It is frequently urged by managers that it is unfair to saddle their operating costs with an amount which they cannot, beyond certain limits, control, as its inclusion makes unfavourable comparison with the costs of another mine in which the tonnage development per foot is higher, and therefore the total operating cost appears less. If the various company reports be scrutinized it will be found that great variations in this charge occur, as, to take two widely different cases, a charge of 5*s.* per ton, included in the French Rand Gold Mine Company's cost, and, in contrast, a charge of only 1*s.* 2*d.* per ton, included by the Geldenhuis Estate Gold Mine Company. There is something to be said for the manager's argument; there is a great deal more to be said on the principle of comparison of costs, which the writer refers to in another place.

The principle of the mine development redemption charge as carried out to-day is unquestionably correct if properly adhered to, but there is no doubt that that principle is very open to, and is not infrequently subject to, abuse.

The temptation to keep costs low, conformably with good returns, is naturally a strong one, and in those cases in which the shoots of ore are very narrow, and the percentage of payable ore to the total ore developed very small—and in consequence the proper charge for mine development redemption very high—the manager is tempted to return a certain tonnage of doubtfully payable ore to meet the heavy expenditure that would otherwise appear against the lesser tonnage of actually payable ore, were that only returned. On various occasions the entire omission of the charge has come before the writer's notice. One in particular was that of a certain property which had been opened up by two shafts and developed quite considerably. The assay plans clearly showed one well-defined narrow payable shoot, the rest of the ground being unpayable. The company started milling, and worked profitably while the developed ore in the narrow shoot lasted. No mine development redemption charge was made in the working costs. On the evidence of the company's assay plans—which proved to be accurate—mine development redemption should have been charged up at 25*s.* per ton. But what manager or company would disclose such a charge? It would simply amount to a confession, on the average Rand mine, of a heavy loss on the operations.

In this particular case the disclosure was never made, but nothing could avert the inevitable failure which ensued.

It is not probable that the charge included in working costs will—excepting in very isolated cases—prove ultimately to have been excessive. The tendency is naturally to keep it as low as possible. If, however, too small a charge be made it simply means that either a large amount must periodically be written off, or, when the mine is exhausted, there will still be a considerable amount standing to the credit of development for which there will be no material value.

In the ordinary way, however, the amount appearing as the charge for mine development redemption is adjusted from time to time, as the conditions demand, and may be taken to represent what is, in the opinion of the administration of the company concerned, the cost of development per ton milled.

CHAPTER XI.

EXPENDITURE.

Working Costs in Deep-Level Mines.

THE actual working costs in deep-level mines, considered apart from interest, will differ from those of the outcrop mines only under the head of mining charges, which include pumping, hoisting, and maintenance. The hoisting charges will be much heavier than those of the outcrop companies, firstly, because of the long and heavy lift, and secondly, because all the working staff must be lowered into and lifted out of the mine by the hoisting engine. The latter must necessarily be a heavy item, considering that the underground labour force required to work a mine running 200 stamps, will be approximately 2500 natives, and 200 white men.

The cost to lower and hoist this force per day, may be taken as the horse power required to deal with the load, plus the cost of engine driving, boiler attendance and maintenance.

Figuring these items out on liberal margins, it would appear that the cost to raise men from a 3000 feet vertical shaft, will entail a monthly expenditure of about £750, equivalent to about 2·2 pence per individual per diem, or to 0·54 shillings per ton crushed, on a 200-stamp milling basis, and crushing 28,000 tons per month.

The charge for hoisting 1200 tons of ore per day, in 5-ton loads, from a depth of 3000 feet, at an average speed of 1500 feet per minute, and assuming one engine to be employed all the time, winding on the tail-rope system, and including engine drivers, firemen, loading and banksmen, figures out at about £2200 per month, equal to 1·6 shillings per ton milled.

The cost for raising water from the mine depends upon the system of pumping adopted, but in any event, under the most economical pumping system, the cost to handle 10,000 gallons an hour against a pressure of approximately 1500 lbs. per square inch, would not be less than £1050 per month, equivalent, on a 200-stamp basis to 0·78 shillings per ton milled. The total cost for pumping and hauling is therefore:—

Hoisting	2·14 shillings
Pumping	0·78 „
Total	2·92 shillings per ton milled.

The average cost per ton milled for hoisting and pumping on outcrop mines with milling installations of 60 to 100 stamps is about 1*s.* 6*d.* On an equivalent installation the costs, as figured above, for deep-level work would be approximately 5·84 shillings per ton milled, or nearly four times the average of the outcrop companies' cost. The bigger installation, however, reduces the cost to say 3*s.* per ton milled, which is what it might be expected to prove in practice to be.

Should hoisting be performed without tail ropes, the cost will be immensely increased, and on the other hand, should it be found practicable to handle a fair proportion of the water by means of bailers, the cost of pumping will be materially diminished.

Expenditure by Deep-Level Companies by end of 1906.

The probable expenditure upon deep-level properties as at December, 1906, may be estimated, in the writer's opinion, at the following amounts:—

Expenditure by companies that will be *producing* at that period	£10,000,000
Expenditure by companies that will be *developing* at that period	3,500,000
Total estimated expenditure	£13,500,000

Analysis of Expenditure by Deep-Level Companies.

An analysis of the amounts spent in bringing present producing companies to the milling stage will show the main items to be approximately as under:—

AVERAGE DEPTH OF SHAFTS, 1100 FEET.

Expenditure Analysis.

Shaft sinking	12 per cent.
Development	33 „
Machinery and Plant	42 „
Buildings	12 „
Sundry expenditure	1 „

It may be remarked here that the relative proportions of expenditure will be greatly changed in the future. The deep-level companies, so far equipped, have had a comparatively small expenditure upon shaft sinking, compared with what will have to be borne by companies which sink shafts say 4000 feet in depth. What concerns us at present, however, is the saving which would have been effected by the present producing companies on the cost of machinery and plant, buildings, construction work, etc., had freight and railage rates been lower, and prospectively what future companies will save in respect of these.

An analysis of the costs under the heads of machinery and plant, and buildings, would give approximately the following proportions, namely :—

Materials	65 per cent.
Labour	30 „
Other expenses	5 „

We have elsewhere seen that the freight and railage rates bear to the total cost of materials, as laid down on the mines, a proportion of 30 per cent. Materials form to the total expense of developing and equipping a deep-level property a proportion of 35 per cent. The cost of freight and railage bears to the total cost, therefore, a proportion of say 10 per cent. If, as is believed to be probable, a reduction on the all-round transport charges is made of 50 per cent., the companies will save 5 per cent. of the total expenditure. This would mean, taking it that the present producing companies have spent on the average £300,000 on surface equipment, a saving of £15,000.

On the next page some particulars are given (in tabular form) of the *producing* deep-level mines, which may be read in conjunction with the following :—

Average depth of shafts, 1100 feet, nearly.

Average cost per stamp, all charges	£4609
„ „ „ development including shaft	£1983
„ „ „ development only	£1454
„ „ „ machinery and plant	£1929
„ „ „ buildings	£453
Total tons crushed as at December 31st, 1898	1,603,021
Total value of gold produced, December 31st, 1898	£23,211,555
Total profit realized, December 31st, 1898	£1,222,727

VARIOUS PARTICULARS OF THE PRODUCING DEEP-LEVEL MINES.

Name of company.	No. of stamps.	No. of claims.	No. of shafts	Depth of vertical shaft.	Total capital expended.	Expenditure per stamp.	Expenditure per claim.	Percentage sorting.	Reefs working.
					£	£	£		
Glen Deep	100	183	2	No. 1 920 No. 2 1103	459,924	4599	2513	30	95 per cent. North, 5 per cent. Main.
Rose Deep	200	181	2	No. 1 840 No. 2 615	678,096	3395	3746	19	North, South, and Main.
Geldenhuis Deep	200	212	2	No. 1 528 No. 2 790	510,120	2550	2406	16	South and Middle.
Jumpers Deep	100	257	2	No. 1 978 No. 2 1352	646,975	6469	2517	33	South and Middle.
Nourse Deep	100	258	3	No. 1 991 No. 2 1492 No. 3 1440	638,877	6388	2476	21	97 per cent. South, 3 per cent. Middle.
Ferreira Deep	100	142	2	No. 1 1204 No. 2 1176	448,523	4485	3158	33	South.
Crown Deep	200	170	2	No. 1 980 No. 2 1250	813,131	4065	4783	17	South and Main Reef Leader.
Langlaagte Deep	100	184	2	No. 1 1270 No. 2 930	656.879	6568	3570	20	South and Main Reef Leader.
Durban Deep	60	232	2	No. 1 1608 No. 2 1409	459,851	4592	1982	46	90 per cent. South.
					£5,312 376				

Cost of Gold Production.

A comparison of the working costs of certain Witwatersrand mines from year to year gives a most instructive lesson with regard to the reduction which better conditions, more experienced administration, and larger scale working can effect.

In the early stages of the industry on these fields—with the difficulties of untrained black labour, indifferent white labour, remote position from seaport, peculiar features of gold deposit, and want of capital to contend against—it is not surprising that working costs were high; but as a set-off to this, the reef was, in the upper oxidized zones—for causes relating to natural concentration—of higher grade, and easier of treatment than when that depth was reached, where the oxidizing influence no longer penetrated, and the dissociation of gold and iron sulphide had to be artificially effected. The impetus given to the development of these fields by the chance feature of high values in the superficial zones of the reef cannot be over-estimated, as otherwise the difficulties enumerated above would, in their reflection in operating costs, have absorbed more than the gold yield, to the discouragement of enterprise and, *ergo*, damage to the fields.

The construction of the railway from the Cape Colony through the then Orange Free State, and its ultimate completion for traffic to these gold fields, marked an epoch in the history of the industry. Prior to this the whole of the materials required for the mines, was transported by means of ox-waggons from the nearest railway terminus, either in Natal or the Cape Colony, and the difficulties experienced in carrying heavy machinery for long distances over unmade roads, in the wet season, could only be met by high rates of transport, and a very slow rate of progress.

The opening of the railway put the Witwatersrand in direct touch with the coast ports of the Cape Colony, and although the railway rates remained abnormally high, the facilities which were obtained by through railway carriage were of infinite value to the fields.

There is a strong tendency in the mind of the average director of mining companies to regard the *working cost per ton* as the sign and seal of good management, without reference to the complicated questions which arise in the settlement of the line of policy best adapted to make a given mine yield its maximum profit per claim. It has been the writer's duty, in his professional capacity, on more than one occasion to condemn a system in vogue in a mine, which, being calculated to ensure to the company a minimum working cost per ton, entirely lost sight of the vital issue of *maximum profit per claim.* In one instance, the mine was working at low cost at a profit of 14*s.* per ton, and after examination of the mine the writer proposed a change in the policy, which would raise the working cost at least 5*s.* per ton, but with the profit of 22*s.* per ton, or a net gain of 8*s.* per ton profit, and this policy was successfully put into practice.

It is urged by some that the lower profit ensures a greater tonnage per claim, and a longer life, and is therefore in the end a benefit to the company. This, however, is an erroneous view, as the figures given below taken from a report by the writer on a Rand mine will prove.

The case in point had reference to the advisability, or otherwise, of carrying very wide stopes on the South Reef, merely because the reef was of great thickness. The writer maintained that the result achieved under the system then in vogue, of mining the total width of ore, was quantity, not quality. The stope averaged approximately 10 feet in width over the two faces, and sampling demonstrated the fact that only the footwall section of reef carried payable gold. To more fully illustrate the point, the actual stope assays, at intervals of 20 feet, are given at the top of the opposite page.

It will be seen that in each instance the footwall section shows highly payable results, whilst the middle and top sections are of very low value. If, instead of taking an average stope of 10 feet in thickness, one of 4 feet had been taken, including only the footwall-section assays as the basis of calculation, the average result would figure out as follows :—

30	inches	footwall	leader	assays	23·5	dwts.	=	705	assay	inches
31	„	„	„	„	21·0	„	=	651	„	„
27	„	„	„	„	23·0	„	=	621	„	„
12	„	„	„	„	2·0	„	=	24	„	„
24	„	„	„	„	65·0	„	=	1560	„	„
16	„	„	„	„	44·0	„	=	704	„	„
19	„	„	„	„	74·0	„	=	1406	„	„
159								5671		

$$\frac{5671}{159} = 35{\cdot}5 \text{ dwts.}$$

Average value 22·7 inches assay, 35·5 dwts.
Average value over 48 inches stope = 16·7 dwts.

COMPARATIVE RESULT.

Stope	122	inches in thickness.	Average assay	8·4	dwts.	per ton.
„	48	„	„ „	16·7	„	„

We now come to the question of the total profit per claim realizable by these methods. Figuring on the assay results

Width.		Assay.	Assay inches.	Average value on total width.
144 inches	30 footwall	23·5	705·0	6·6 dwts.
	18 ,,	waste	—	
	42 ,,	4·5	189·0	
	12 ,,	waste	—	
	42 ,,	1·5	63·0	
147 inches	31 footwall	21·0	651·0	5·4 dwts.
	33 ,,	waste	—	
	36 ,,	nil	—	
	14 ,,	waste	—	
	33 ,,	4·5	148·5	
145 inches	27 footwall	23·0	621·0	7·4 dwts.
	23 ,,	waste	—	
	41 ,,	5·0	205·0	
	8 ,,	nil	—	
	46 ,,	5·5	233·0	
144 inches	12 footwall	2·0	24·0	1·8 dwts.
	48 ,,	waste	—	
	48 ,,	1·5	72·0	
	36 ,,	4·5	162·0	
90 inches	24 footwall	65·0	1560·0	17·7 dwts.
	30 ,,	waste	—	
	36 ,,	1·0	36·0	
90 inches	16 footwall	44·0	704·0	9·4 dwts.
	24 ,,	waste	—	
	41 ,,	3·5	143·0	
	9 ,,	waste	—	
94 inches	19 footwall	74·0	1406·0	17·7 dwts.
	38 ,,	waste	—	
	19 ,,	8·5	161·5	
	3 ,,	waste	—	
	15 ,,	7·0	105·0	

Average of stope, 122 inches. Average of assay, 8·4 dwts.

above given, and calculating on sorting out 15 per cent., the recovery of gold from an assay value of 8·4 dwts. would be 25*s.* per ton. On the same basis the gold recovery from an assay of 16·7 dwts. would be 49*s.* per ton. The comparative profits to be made from say one claim, would then be as under:—

Thickness of reef.	Tons per claim less sorting.	Yield per ton.	Cost per ton.	Profit per ton.	Profit per claim.		
		s.	*s.*	*s.*	£	*s.*	*d.*
10 feet	50,000	25	17	8	20,000	0	0
4 ,,	19,500	49	21	28	27,300	0	0

Difference per claim in favour of narrower stope £7300, without considering the increased value of the total profit, because of the much shorter period taken for realization.

The object in citing these instances is to show how great a fallacy it is to regard low working costs as the fundamental principle of skill in the management of mining affairs. The writer does not mean to imply that every care and precaution should not be exercised in expenditure; on the contrary, he insists that such expenditure shall be so directed that the object for which the company exists shall be as far as possible fulfilled —and that is, the highest profit which can be made, and in the shortest possible time, reconcilable with an intelligent plan of operations. A sentimental regard to public opinion on the question of working costs is the last thing that should be present in the mind of an engineer, when framing a policy to suit the conditions presented by a given mining enterprise, as it is evident that the financial basis of such an opinion is grounded on erroneous premises.

Following on the subject of low working costs, comes the no less palpably illogical rule of comparing the costs of one mine with another. These are, in fact, no more reasonably comparable than are the yields from various mines, situated in different sections, and with quite different problems to elucidate. The differences that arise may be any or all of the following: reef width; percentage of sorting; hardness of country rock; quality and size of equipment; facilities for labour and materials; handling of ore in mine (this being a most important item, as in one mine there may be several distinct sections caused by dykes and faults, or positions of claim blocks, all requiring separate treatment, and manifestly at increased cost); compactness of plant arrangement, etc.

If the variation in operating cost that these differences can introduce be for one moment considered, it will be granted that only those mines working under similar natural and artificial conditions should be compared, and none other. The operating cost of every mine must be studied to suit the special case, and that without reference to the standard of cost maintained by another quite differently conditioned mine, whether adjacent or at a distance. Every present feature should be taken into account when figuring on a basis of working costs, and the policy should be, to sift details and propose economies, with a full appreciation of the position, and not—as is so generally the rule—to make unfavourable comparisons and sweeping generalizations, in cases which present, for this matter, only an approximate parallelism.

The operating costs of the Witwatersrand mines are in most instances given in detail in reports issued by the companies' secretaries at monthly periods. Subjoined are two examples, the one by the Van Ryn Gold Mines Estate, and the other by the Meyer and Charlton Gold Mines Company, Limited, both of which are included in the financial group represented by the writer.

THE VAN RYN GOLD MINES ESTATE, LIMITED.

Report for month of July, 1890.

MINE.

Number of feet driven, sunk, and risen		2,163 feet.
Quartz mined	25,712 tons.	
Less quartz rock discarded	6,696 tons.	
Quartz sent to mill and crushed		19,016 tons.

MILL.

Number of days working (125 stamps)	30 days.
Ore crushed	19,016 tons.
Yield in smelted gold	6113·95 ounces.
Yield per ton	6·430 dwts.

CYANIDE WORKS.

Tailings treated	13,440 tons.
Yield in bullion, at 60*s.* per ounce	2871·3 ounces.
Yield per ton treated	4·272 dwts.
Yield per ton (on basis of tonnage milled)	5·019 dwts.
Working cost per ton treated	2*s.* 5·598*d.*

EXPENDITURE AND REVENUE.

On basis of tonnage milled	19,016 tons.

WORKING EXPENDITURE.

	Working cost.			Cost per ton.	
	£	*s.*	*d.*	*s.*	*d.*
Mining	6,399	6	2	6	8·765
Milling	2,784	10	6	2	11·143
Maintenance	1,230	14	3	1	3·533
Gen. charges (mine)	945	9	10	0	11·933
Mine development redemption	3,803	4	0	4	0·000
Cyaniding	1,657	10	0	1	8·919
Fixed charges (including licences, insurances, etc.	600	0	0	0	7·573
	£17,420	14	9	18	3·866
Profit for month	14,017	17	1	14	8·919
	£31,438	11	10	33	0·785

REVENUE.

	Value.			Value per ton.	
	£	s.	d.	s.	d.
Gold from mill, 6113·95 ounces at 74s. 3·36d. per ounce	22,707	5	0	23	10·587
Gold from cyaniding 2871·3 ounces, at 60s. per ounce	8,556	6	10	8	11·989
Sundry revenue	175	0	0	0	3·209
	£31,438	11	10	33	0·785

EXPENDITURE ON CAPITAL ACCOUNT.

	£	s.	d.
Development	5232	16	6
Permanent works	2503	19	2
Machinery and plant	143	1	11
Cyanide works	167	13	4
Buildings	37	18	8
Reservoirs and dams	303	8	5
Live stock	193	19	3
	£8582	17	3
Less Mine development redemption	3803	4	0
	£4779	13	3

The expenditure of the various departments is again analyzed into items, and, as an illustration, the details of the New Goch development and mining costs are appended.

NEW GOCH GOLD MINES, LIMITED.

ANALYSIS OF DEVELOPMENT COSTS.

May, 1899.

	£	s.	d.	£	s.	d.
Proportion of staff salaries	70	0	0			
Rock drill boss	32	10	0			
Rock drill fitter	27	0	0			
One-third timberman's salary	19	10	0			
Men sinking winze	63	0	0			
One engine-driver on compressor	31	0	0			
Maintenance work for diamond drill	2	16	0			
Fitter's time	11	4	0			
Smith's time	3	0	0			
				260	0	0
Contractors in mine	1078	4	9			
Contractors sharpening drills	112	0	9			
				1190	5	6
Dynamite	1065	13	6			
Cartage on same	15	0	0			
Fuse	22	15	0			
Detonators	11	12	9			
				1115	1	3
Coal for compressor and shafts				335	15	6
Rock drill steel	11	3	10			
Smithy coal	11	3	8			
				22	7	6
Rock drill spares				61	0	6
Piping and fittings				35	1	11
Sundry stores				16	7	3
Oils and candles				51	15	11
Natives labour and food				549	12	6
				£3637	7	10

ANALYSIS OF WORKING COSTS.

STOPING.

	£	*s.*	*d.*	£	*s.*	*d.*
European labour	375	10	0			
Native labour and food	1212	9	8			
Dynamite	701	0	6			
Fuse	41	10	0			
Caps	20	16	6			
Candles	91	0	4			
Drill steel	55	11	1			
Smithy coal	8	16	0			
Sharpening drills	83	10	9			
Shovels, hammers, etc.	2	16	6			
Cartage dynamite	7	10	0			
Assay account	16	19	1			
				2617	10	5

(Cost per ton milled, 6*s.* 0·58*d.*)

PUMPING AND HAULING.

	£	*s.*	*d.*			
European labour	272	7	3			
Native labour and food	80	2	5			
Coal	276	13	9			
Oils, etc.	30	4	10			
Waste	2	6	6			
Carbonate of soda	8	2	0			
Cartage coal	38	9	6			
				708	6	3

(Cost per ton milled, 1*s.* 7·64*d.*)

TRAMMING.

	£	*s.*	*d.*			
European labour	65	5	0			
Native labour and food	249	15	1			
Forage, chaff, etc.	13	18	8			
				328	18	9

(Cost per ton milled, 9·12*d.*)

PLATELAYING AND TIMBERING.

	£	*s.*	*d.*			
European labour	68	10	0			
Native labour and food	25	9	9			
Deals	36	15	3			
Mine poles	68	8	0			
Rails and sleepers	9	18	4			
				209	1	4

(Cost per ton milled, 5·80*d.*)

MAINTENANCE.

	£	*s.*	*d.*			
European labour	384	18	6			
Native labour and food	136	15	3			
Piping and pump fittings	52	15	9			
Bar iron and steel	23	19	2			
Packing	3	9	1			
Cement and lime	5	12	6			
Fire bricks	3	16	0			
Sundry stores	48	2	4			
Castings	31	7	4			
Boiler tubes	103	3	11			
Electric fittings	8	5	2			
Quarter share cost new cages	52	0	0			
Wire and Manilla ropes	58	2	11			
White metal	7	6	3			
				919	14	2

(Cost per ton milled, 2*s.* 1·50*d.*)

GENERAL CHARGES.

	£	*s.*	*d.*			
Manager and staff salaries	435	9	9			
Compound native wages and food	116	9	7			
Sundry cash items	67	11	6			
Medicines and sanitary fees	33	10	6			
Stationery and tickets	21	16	6			
				674	17	10
Total				£5458	8	9

(Cost per ton milled, 1*s.* 6·72*d.*)

(Total cost per ton milled, 12*s.* 7·36*d.*)

SUMMARY.

	Total cost.			Cost per ton.	
	£	*s.*	*d.*	*s.*	*d.*
Stoping	2617	10	5	6	0·58
Pumping and hauling	708	6	3	1	7·64
Tramming	328	18	9	0	9·12
Platelaying and timbering	209	1	4	0	5·80
Maintenance	919	14	2	2	1·50
General charges	674	17	10	1	6·72
	£5458	8	9	12	7·36

8655 tons milled.

The total expenditure is analyzed as on the next page, taking the Meyer and Charlton Mine as an instance :—

Analysis of Working Expenses for Twelve Months.	Working costs.			Development costs.		Total working and development costs.		
	Cost.	Cost per ton.	Per cent.	Cost.	Cost per ton.	Cost.	Cost per ton.	Per cent.
	£ s. d.	s. d.		£ s. d.	s. d.	£ s. d.	s. d.	
Salaries and white wages	23,877 6 5	4 3·809	28·754	8,009 13 2	1 5·380	31,886 19 7	5 9·189	31·673
Native wages	19,212 15 11	3 5·688	23·137	2,053 10 11	0 4·456	21,266 6 10	3 10·144	21·124
Native food	3,372 12 1	0 7·318	4·061	394 3 3	0 0·855	3,766 15 4	0 8·173	3·741
Coal	7,542 14 3	1 4·367	9·083	1,465 16 7	0 3·180	9,008 10 10	1 7·547	10·399
Dynamite	5,968 18 2	1 0·952	7·188	4,500 7 1	0 9·765	10,469 5 3	1 10·717	1 158
Candles	977 3 11	0 2·120	1·176	188 12 10	0 0·409	1,165 16 9	0 2·529	1·158
Coke and gas carbon	153 2 2	0 0·333	0·185			153 2 2	0 0·333	0·153
Cyanide	3,841 14 7	0 8·335	4·626			3,841 14 7	0 8·335	3·816
Chemicals [auditors' fee	630 19 3	0 1·369	0·759			630 19 3	0 1·369	0·627
Directors, London and Berlin committees, and	2,405 0 0	0 5·219	2·896			2,405 0 0	0 5·219	2·382
Electric spares	103 13 2	0 0·225	0·124			103 13 2	0 0·225	0·103
Forage, chaff, and bran	218 1 0	0 0·473	0·262			218 1 0	0 0·473	0·217
Fuse and detonators	976 9 0	0 1·474	0·818	159 9 0	0 0·346	838 18 0	0 1·820	0·833
Hospital medicines, etc.	422 18 9	0 0·917	0·509			422 18 9	0 0·917	0·420
Insurance, fire and life	623 19 9	0 1·354	0·752			623 19 9	0 1·354	0·620
Interest and commission	12 17 6	0 0·028	0·016			12 17 6	0 0·028	0·013
Lubricants	781 12 9	0 1·696	0·941	214 15 10	0 0·466	996 8 7	0 2·162	0·990
Licences	85 5 0	0 0·185	0·103			85 8 0	0 0·185	0·085
Mercury [stems, etc.	109 12 0	0 0·238	0·132			109 12 0	0 0·238	0·109
Mill spares—shoes, dies, cams, cam shafts,	1,910 7 6	0 4·145	2·300			1,910 7 6	0 4·145	1·898
Pipes and pipe fittings	62 17 5	0 0·136	0·076	67 5 0	0 0·146	130 2 5	0 0·282	0·129
Rope, steel, and Manilla	35 2 5	0 0·076	0·042			35 2 5	0 0·076	0·034
Rock drill spares	15 16 6	0 0·034	0·019	271 2 1	0 0·588	286 18 7	0 0·622	0·285
Steel [advertising	670 19 6	0 1·456	0·808	200 1 4	0 0·434	871 0 10	0 1·890	0·865
Stationery—postages, telegrams, printing and	1,026 8 8	0 2·227	1·236			1,026 8 8	0 2·227	1·019
Sundry stores—bar-iron, bolt nuts, etc.	3,363 5 3	0 7·298	4·051	114 14 6	0 0·249	3,477 19 9	0 7·547	3·455
Sanitation fees	578 6 6	0 1·255	0·697			578 6 6	0 1·255	0·575
Sundries	2,764 18 11	0 5·999	3·330			2,764 18 11	0 5·999	2·746
Timber (mining)	623 17 7	0 1·353	0·759			623 17 7	0 1·353	0·619
Timber (deals, etc.)	390 4 8	0 0·847	0·470			390 4 8	0 0·847	0·388
Zinc	573 4 9	0 1·244	0·690			573 4 9	0 1·244	0 569
	83,035 5 4	15 0·170	100·000	17,639 11 7	3 2·274	100,674 16 11	18 2·444	100·000
Mine development redemption	16,591 7 0	3 0·000						
	99,626 12 4	18 0·170 working costs for one year.						

CHAPTER XII.

FORECAST OF POSITION OF DEEP-LEVEL MINES IN 1906.

Estimated Number of Stamps at End of 1906.

The total number of stamps now in operation by deep-level companies is 1370. These are all in the first row of deeps. At the end of the year 1906, it is safe to assume that all of the first and second row of deep-levels, that started active operation before the war, will have reached the producing stage, and in the mean time some of the first-row mines will have increased their plant.

In all, there should be at the end of the period named at least 18 new deep-level companies producing, with an average milling capacity of say 100 stamps—this being actually a minimum figure—making a total of 1800, whilst the present producing deep-level companies will increase their milling power by not less than 500 stamps. To this total must be added the additions which the bigger outcrop companies will make in the same period. The East Rand Proprietary Mines will probably be the largest individual contributor, totalling at least 400 stamps. Other companies to increase their milling capacity within our limits will be (1) Witwatersrand; (2) Simmer and Jack; (3) New Goch; (4) Wolhuter—totalling not less than 400 stamps. In the extreme western section of the Rand, the Randfontein group of mines will largely increase their present capacity by extending plant, and probably will have brought new companies to the producing stage, bringing in all, with other companies in the Krugersdorp district, not less than 700 stamps to swell the total. On the extreme east from Boksburg, east to Modderfontein, there will not be less than an additional 200 stamps.

The estimated total is then as follows :—

From new deep-levels	1800	stamps
Additions to capacity of present deep-level companies . .	500	„
East Rand proprietary mines	400	„
Central section mines	400	„
Western section mines	700	„
Eastern section mines	200	„
From new deep-level mines not yet started operations . .	500	„
Estimated additional stamps to end 1906	4500	„
Average number of stamps running prior to war . . .	5762	„
Total,	10,262	„

Tonnage crushed per Month at End of 1906.

The average tonnage crushed per month on the whole Witwatersrand, for the first eight months of 1899 preceding the war, was 758,415 tons, from a total average number of 5762 stamps. The mills now erecting, and those that will be erected before the end of the period named, will be of heavy pattern, not less than 1250 lbs. per stamp. The average duty of such mills will be about 5 tons per stamp per diem, working with 600-mesh screen, and this figure we may take as the stamp duty factor. Assuming the average running time of the mill as 28 days per month, and taking the estimated additional 4500 stamps as doing 5 tons per stamp per day, the total tonnage which they will require is 4500 × 5 × 28 = 630,000 tons per month. To this total add the requirements of the 5762 stamps working before the war, and the total ore milling per month on Witwatersrand, at the date named, will reach the colossal figure of 1,688,415 tons.

The quantity of ore that will be mined to supply this requirement will probably be at least 20 per cent. more than the tonnage milled, the balance being represented by the waste rock discarded on the sorting floors, so that the total tons of broken rock hoisted to the surface will be $\frac{1,688,415 \times 100}{80} = 2,110,520$.

Probable Gold Output of the Witwatersrand at End of 1906.

The average declaration of the gold output on these Fields for years 1898 and 1899 was 6·41 dwts. per ton caught on the plates, and 4·07 dwts. per ton extracted from the sands by the cyanide process, and 2·01 dwts. extracted from slimes. Only comparatively few of the outcrop companies are treating slimes, but it is safe to assume that the majority, if not all, of the companies now equipping, and those that have yet to equip, will adopt some method of slimes treatment. We may calculate, therefore, that of the total stamps employed in ore reduction on the Rand at the end of 1906, not less than 5000 will have the slimes process as part of the general treatment scheme. This will increase the average return of gold, for those companies employing this latter process, by half a dwt. per ton. The average value of the gold won per ton of ore milled in the 9 months preceding the war was 41*s.* 3*d.*

Taking as an average the figures thus supplied, and basing our calculations on them, we are able to estimate that at the end of 1906 there will be a total of 1,688,415 tons of ore milled per month, for a gross yield of say 541,137 ounces fine; 1,181,890 tons of tailings treated for a gross yield of 240,514 ounces fine; and 210,000 tons of slimes treated for a gross yield of 21,105 ounces fine: the aggregate grand total being 812,756 ounces fine valued at £3,450,828, or an average extraction of 40*s.* 9*d.* per ton milled.

Taking into consideration the economies, which under the new regime it is confidently expected will be introduced, and the better administration and improved appliances which the mines will have by that time secured, the writer does not consider it unduly optimistic to assume that at the date named a substantial additional profit per ton will be earned.

CHAPTER XIII.

VALUATION OF MINES.

Mine Valuation.

The special features which characterize the Rand banket deposits make it possible to introduce computing systems, relative to their extent and value, which, intelligently applied, have been abundantly proved to strike well within the limits of practical accuracy. The lapse of time with its concomitant experience, and the application of rule, have crystallized into an axiom the enunciation, that within a given area—taking especially the direction of the ore shoots of the section into consideration—the values of unworked portions may be closely calculated from the results achieved in adjoining developed mines. The importance of such a generalization with respect to the development of deep-level areas—the value of which is otherwise purely conjectural—and the stimulus which is thereby supplied for enterprise in that direction, must be at once admitted, on the evidence afforded in the cases of the majority of the deep-level companies now in active operation, which have commenced work upon the assumption that the enunciation previously cited would have a satisfactory demonstration.

Despite, however, these special features, which are fundamentally necessary to the establishment of the generalization indicated, they have only contributed the bases of the proposition, and are, if faultily construed, capable of supporting a deduction which would stifle all further enterprise. This was exemplified in the earlier stages of many now successful outcrop companies, before the policy of extensive general development was recognized as the element required, in combination with the bases aforesaid, to ensure commercial success.

Once having obtained the data for generalization, and having derived therefrom a satisfactory indication of the value of a property which it is proposed to work, the functions of the generalization cease, because so soon as development begins upon the reefs we change the methods to particulars, and each mine is valued upon certain systems which are employed for the purpose.

The first step in the valuation of a developed mine is the practice of "sampling" it—that is, to extract at carefully measured intervals, along all drives and winzes, a section of the reef for assay. Simple as it may appear to do this, it is really a most arduous operation, and one entailing more than average intelligence and conscientiousness to properly perform. The tendency in all sampling work is towards selection, not only in respect of the richer sections of the ore body, but equally in respect to the conditions of the reef at the points to be sampled —whether hard or soft; whether easily or with difficulty accessible—and it is only by the exercise of great caution in the first case, and determination in the second, that these tendencies can be overcome.

A sampler should at the same time, if the importance of his work is fully recognized, be a trained scientific observer, noting, as development, and *pari passu* the sampling, proceeds, every feature of the reef, whether the pay section is always exposed; every fault and dyke, their nature and effect. Such observations, if systematically recorded, assist greatly in the calculation of general tonnage values, and materially assist in choosing the positions of stopes to the very best advantage regarding faults, which latter, if not carefully considered and provided against, may easily so dislocate the reef in a stope, that attack from another quarter is necessitated.

The sampler's first work is to measure off the intervals at which sections of the reef are to be taken. The intervals are generally either 5 feet or 10 feet, depending upon the width of the pay section; if this be narrow the intervals are never greater than 5 feet, because in narrower reefs there is liable to be more violent fluctuation in value than in those of average stoping width, and for that reason it is important to minimize the possibility of high individual values having an undue effect upon the general average. In reefs of greater width, on which samples are taken at 10-foot intervals, the writer's practice is to temporarily exclude results which depart greatly from the average of the samples taken in the vicinity, until the total value of the assays round the developed block have been obtained, when the excluded results before mentioned are included as portions of the general result. The rationale of this method is to guard against the danger of enriching short sections of the reef by the inclusion of values which, whilst perfectly correct in themselves, will give erroneous results if averaged in with the adjoining sections. This can be illustrated in the following manner:—

A longitudinal section of reef is represented in Fig. 34, showing the width and assay values of samples obtained at 10-foot intervals.. The average value of each section is arrived at by multiplying the width into the assay, the product being "assay inches." The sum of the widths then divided into the assay inches gives the average value, and the total width divided by the number of samples taken gives the average width sampled.

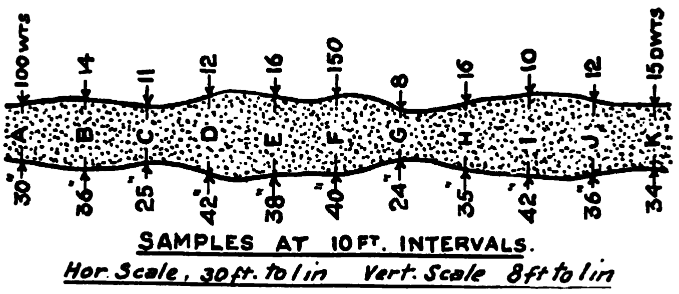

10 DWTS
14
11
12
16
150
8
16
10
12
15 DWTS
A
B
C
D
E
F
G
H
I
J
K
30"
36"
25"
42"
38"
40"
24"
35"
42"
36"
34"
SAMPLES AT 10FT. INTERVALS.
Hor. Scale, 30ft. to 1 in Vert. Scale 8ft to 1 in

Fig. 34.

In the case of the foregoing samples, the average result would be as follows:—

Sample No.	Width Sampled.	Assay Value.	Assay Inches.
A	30	10	300
B	36	14	504
C	25	11	275
D	42	12	504
E	38	16	608
F	40	150	6000
G	24	8	192
H	35	16	560
I	42	10	420
J	36	12	432
K	34	15	510
	382		10305

Result—

$$\left.\begin{array}{l} \dfrac{10305}{382} = 27 \text{ dwts. nearly} \\ \dfrac{382}{11} = 34{\cdot}7 \text{ inches} \end{array}\right\} 34{\cdot}7 \text{ inches average } 27 \text{ dwts.}$$

In practice, sample F would be eliminated from the average obtained, on account of its abnormal value. The result would then be—

$382 - 40 = 342$ total inches
$10305 - 6000 = 4305$ total assay inches,

then—

$$\left.\begin{array}{l} \dfrac{4305}{342} = 12{\cdot}5 \text{ dwts.} \\ \dfrac{342}{10} = 34{\cdot}2 \end{array}\right\} 34{\cdot}2 \text{ inches average } 12{\cdot}5 \text{ dwts.}$$

The inclusion of sample F would, therefore, in this instance, raise the value of each of the five samples on either side of it by no less than 15·7 dwts., more than doubling the actual average which they show when it is eliminated. Equally grave errors may arise if the intervals at which samples are taken be not spaced regularly, that is to say, if the standard of 5 feet or 10 feet sampling, as the case may be, be not rigidly observed.

The writer once examined an assay plan on one of the Witwatersrand mines, which exemplified in a striking manner the errors which this departure from the system can introduce. The said assay plan reflected the values of South Reef Leader, narrow, but in places very rich, on a width of 6 inches. The samples were shown on the plan somewhat as illustrated in Fig. 35.

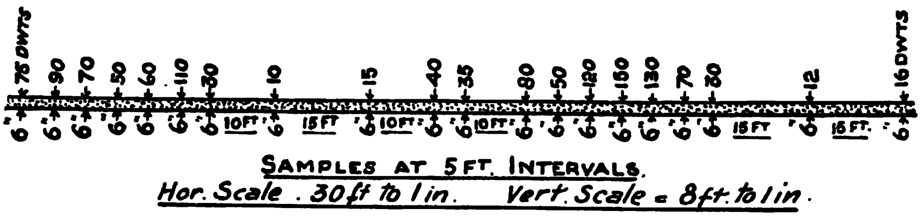


Fig. 35.

In the assay records these results were entered up in rotation, as though each sample represented a 5-foot interval; and the result so obtained was carried down as a result accurately reflecting the total distance sampled, which is, in the illustration above given, 175 feet. The average value of the above sampling, on this system, would be as follows:—

Sampled place.	Width of reef sampled.	Assay value.	Assay feet.
5 feet east of face of drive	6	90	540
10 ,, ,, ,,	6	70	420
15 ,, ,, ,,	6	50	300
20 ,, ,, ,,	6	60	360
25 ,, ,, ,,	6	110	660
30 ,, ,, ,,	6	30	180
45 ,, ,, ,,	6	10	60
65 ,, ,, ,,	6	15	90
80 ,, ,, ,,	6	40	240
90 ,, ,, ,,	6	35	210
105 ,, ,, ,,	6	80	480
110 ,, ,, ,,	6	50	300
115 ,, ,, ,,	6	120	720
120 ,, ,, ,,	6	150	900
125 ,, ,, ,,	6	130	780
130 ,, ,, ,,	6	70	420
135 ,, ,, ,,	6	30	180
155 ,, ,, ,,	6	12	72
175 ,, ,, ,,	6	16	96
	114		7008

$$\frac{7008}{114} = 61{\cdot}5 \text{ dwts.}$$

Result—Average value of reef for 175 feet sampling 6 inches = 61·5 dwts.

This result is quite incorrect, because each sample is taken to represent an equal distance interval, or to be equal factors in a sum of 175 feet of distance, when, as a matter of fact, very widely different proportions of the total length are represented. To properly average the value of the total length sampled, it is necessary to remember that each sample must be taken to indicate the value of the ore as from the sample immediately preceding, and although it is bad practice to have comparatively long intervals, the values of which are only reflected by one assay result, yet such instances do occur, and for the sake of illustration the one under notice has been chosen.

Taking the view that each sample represents the value of an interval, we may now average out the assay results shown in the preceding table thus :—

1st	result,	5	feet	interval,	averages	90	dwts. =	450	assay	feet.
2nd	„	5	„	„	„	70	„	350	„	„
3rd	„	5	„	„	„	50	„	250	„	„
4th	„	5	„	„	„	60	„	300	„	„
5th	„	5	„	„	„	110	„	550	„	„
6th	„	5	„	„	„	30	„	150	„	„
7th	„	15	„	„	„	10	„	150	„	„
8th	„	20	„	„	„	15	„	300	„	„
9th	„	15	„	„	„	40	„	600	„	„
10th	„	10	„	„	„	35	„	350	„	„
11th	„	15	„	„	„	80	„	1200	„	„
12th	„	5	„	„	„	50	„	250	„	„
13th	„	5	„	„	„	120	„	600	„	„
14th	„	5	„	„	„	150	„	750	„	„
15th	„	5	„	„	„	130	„	650	„	„
16th	„	5	„	„	„	70	„	350	„	„
17th	„	5	„	„	„	30	„	150	„	„
18th	„	20	„	„	„	12	„	240	„	„
19th	„	20	„	„	„	16	„	320	„	„
		175						7960		

Result—175 feet sampled contain a product of 7960 assay feet.

$$\text{Average value } \frac{7960}{175} = 45{\cdot}5 \text{ dwts.}$$

By using each interval, therefore, in its true proportion to the whole, we have obtained a result of 45·5 dwts., representing the average value of the total 175 feet sampled, as against 61·5 dwts., when incorrectly assuming each interval to represent an equal proportion of the whole. The practical effect of employing the first method would be to overestimate the value of the ore by 16 dwts. per ton on the width sampled, the seriousness of which it is unnecessary to further dwell upon.

It is plain from the preceding calculations that, if samples are taken at regular intervals, and if the sampled width be invariable, the average of the whole may be obtained by simply dividing the sum of the assay results by the number of intervals represented.

Having considered the importance of sampling at regular intervals, we now come to the question of how best to sample the reef at these intervals so that an essential degree of accuracy may be obtained.

The method of sampling of reefs varies as to the conditions presented in the ore body under valuation. For the purpose of illustration let us take three cases as under :—

(1) Narrow clean reef body.

(2) Reef of medium thickness with intercalated quartzite or "waste" rock.

(3) Wide clean reef body.

(1) The south reef in the Roodepoort district furnishes probably the best instance of a continuously narrow and rich reef. The actual width of the reef in the richest sections is seldom more than two inches, and in sampling such a thin reef body it is customary to include a portion of the quartzite above and a portion below the reef proper, the sample being carefully channeled to a given width and depth, to obviate as far as the danger of including a disproportionate quantity of the rich possible reef. It is a matter for dispute whether this method of including the enclosing quartzite in the sample taken—or whether it should be untouched in the sampling, and only reef proper be taken—is calculated to give better results. From experience, the writer would rather exclude the quartzite, for the simple reason that, being generally tougher than the reef matter, and taking the bias of the sampler likewise into consideration, a strictly proportional quantity as to the whole sample of reef and quartzite is seldom obtained, and thus the assay value is higher than, with accurate proportioning, it should be. The practical effect is similar to that obtained by careless measuring of the widths, which may be thus illustrated.

Supposing a series of samples to be taken on a reef actually 2 inches in width, and the value is reported as upon 3 inches, and assume that the value so reported is 120 dwts. The value of 3 inches at 120 dwts. on a stoping width of 36 inches, is 10 dwts., but the value of two inches at 120 dwts. on the same width is only 6·6 dwts., the result in the one case being payable, and in the other unpayable. Since, therefore, the inclusion of quartzite in the sampling tends to introduce such errors, it is wiser to sample only clean reef matter.

(2) In sampling reef of medium thickness, having intercalated bands of quartzite, the writer invariably has the samples taken from the reef matter only, measuring the quartzite and calculating it in the average result. Let Fig. 36 represent a reef section to be sampled:—

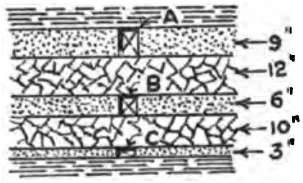


Fig. 36.

At the points A, B, and C sections of the reef would be taken and separately assayed, whilst a note would be made of the measurements of the intervening quartzite. The results would appear for the calculation of average width thus:—

A. Hanging wall section,	9	inches assay, say	8 dwts. =	72 assay inches.
Band of quartzite,	12	,, ,,	nil.	nil.
B. Middle reef section,	6	,, ,, ,,	10 dwts. =	60 assay inches.
Band of quartzite,	10	,, ,,	nil.	nil.
C. Footwall reef section,	3	,, ,, ,,	100 dwts. =	300 assay inches.
Total width	40	,,	Total assay inches	432

Average value $\frac{432}{40}$ = 10·8 dwts. on a width of 40 inches.

(3) In sampling wide clean reef body, it is the writer's practice to divide the width into sections which are separately bagged and assayed. The object aimed at is to avoid as far as possible the liability which is present, when sampling, say a 5-feet reef in one section, of taking unequal proportions of the ore, and thus vitiating the result. A limit of 18 to 20 inches is generally made by the writer as the greatest which may be taken in one section, consequently in a 5-feet reef, at each distance interval, three separate samples would be taken, and the results averaged out over the whole width (see Fig. 37) as under :—

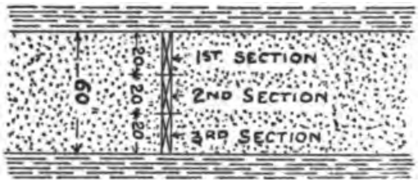


Fig. 37.

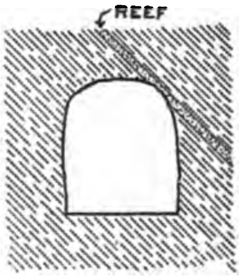


Fig. 38.

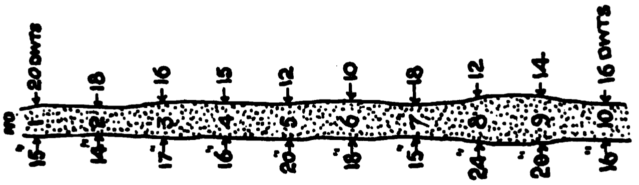
1st section, width	20	inches, assay value, say	8 dwts. =	160	assay inches.
2nd ,, ,,	20	,, ,, ,,	10 ,, =	200	,, ,,
3rd ,, ,,	20	,, ,, ,,	12 ,, =	240	,, ,,
Total width	60	,,	Total assay inches	600	

Average $\frac{600}{60}$ = 10 dwts. on a width of 60 inches.

In all sampling, the greatest care must be taken to measure the reef exposed at right angles to its inclination : otherwise, if a reef be only partially exposed in the hanging wall of a level, it is possible to make very grave errors, as will be understood from the diagram in Fig. 38. In this case a careless sampler could take a section of reef, perhaps 3 feet long, and return it as 3 feet wide, when in reality he would be only skimming the under side of the reef body, and its actual width might not be more than 3 inches.

Standardizing Values on Fixed Widths.

In all cases it is of the utmost importance to fix a width over which the average assay values are to be calculated. This is best done by studying the angle of dip and the attendant bedding conditions of the reef, and determining from these what width of stope it will be practicable to carry. The nearer the reef approximates to verticality, the narrower will it be possible to stope; and conversely, the nearer the reef approaches to horizontality, the more impracticable does narrow stoping become. The general practice on the Rand is to fix a minimum width of 36 inches as a stoping basis, and all assay values on reefs narrower than 36 inches are brought to this standard width thus (Fig. 39):—



SAMPLES AT 10 FT. INTERVALS.
HOR. SCALE, 30 FT. TO 1 IN. VERT. SCALE. 8 FT TO 1 IN.

Fig. 39.

Sample No.	Width.	Assay value.	Assay inches.
1	15	20 dwts.	300
2	14	18 „	252
3	17	16 „	272
4	16	15 „	240
5	20	12 „	240
6	18	10 „	180
7	15	18 „	270
8	24	12 „	288
9	20	14 „	280
10	16	16 „	256
	175		2578

$$\text{Average value } \frac{2578}{175} = 14{\cdot}7 \text{ dwts.}$$

$$\text{Average width } \frac{175}{10} = 17{\cdot}5 \text{ inches.}$$

Result—Average value 14·7 dwts. on 17·5 inches width. Standardizing this result on the fixed width of 36 inches the value becomes: as 36 : 17·5 :: 14·7 = 7·1 dwts.

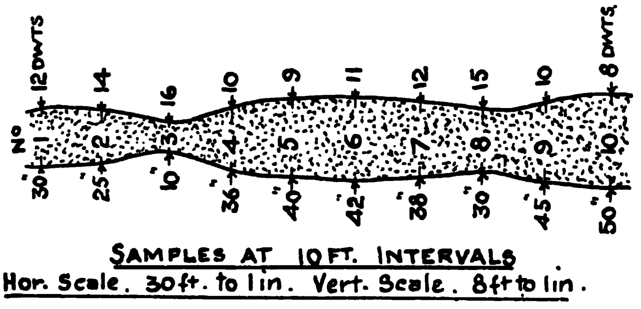
In cases where a reef is very variable in width, falling above and below the standard minimum of 36 inches, the value of each sample should be at once calculated out as upon the said minimum, and the value of samples taken on thicknesses of over 36 inches will remain unaltered, thus (Fig. 40):—

Sample No.	Width.	Assay value.	Value on 86-inch stope.
1	30	12 dwts.	10·0 dwts.
2	25	14 "	9·7 "
3	10	16 "	4·4 "
4	36	10 "	10·0 "
5	40	9 "	9·0 "
6	42	11 "	11·0 "
7	38	12 "	12·0 "
8	30	15 "	12·5 "
9	45	10 "	10·0 "
10	50	8 "	8·0 "
			96·6

Result—

$$\frac{96{\cdot}6}{10} = 9{\cdot}6 \text{ average value on 36 inches.}$$

If the reef averages more than 36 inches in width, then a new standard, say 48 inches, is established, to which all values are reduced; but as each inch of the 48 in a calculated average is of equal value, it matters not—excepting for the calculation of tonnage—whether the value be reckoned on 36 inches or on 48 inches, as an established value on 48 inches must be the same as a value on any of its integers, and therefore the same as upon 36 inches. If, however, the value be established on 36 inches, and it is, say, 10 dwts. for that width, it would only be 7·5 dwts. on 48 inches.


12 DWTS
14
16
10
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8 DWTS
No 1
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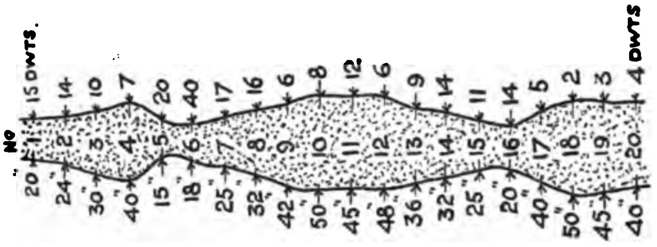
9
10
30"
25"
10"
36"
40"
42"
38"
30"
45"
50"
SAMPLES AT 10 FT. INTERVALS
Hor. Scale. 30ft. to 1in. Vert. Scale. 8ft to 1in.

Fig. 40.

Erroneous Methods of calculating Averages.

It may be well at this point to illustrate, by example, the errors which may arise if erroneous methods of computing averages are introduced in the valuation of the ore reserves of a mine. We shall take as a basis for illustration a length of 100 feet of level sampled at 5 feet intervals (Fig. 41), as under:—



SAMPLES AT 5 FT. INTERVALS.
Hor. Scale = 30 FT. TO 1 IN. Vert. Scale .8 FT. TO 1 IN.

Fig. 41.

Sample No.	Width.	Assay value.	Assay inches.
1	20	15 dwts.	300
2	24	14 ,,	336
3	30	10 ,,	300
4	40	7 ,,	280
5	15	20 ,,	300
6	18	40 ,,	720
7	25	17 ,,	425
8	32	16 ,,	512
9	42	6 ,,	252
10	50	8 ,,	400
11	45	12 ,,	540
12	48	6 ,,	288
13	36	9 ,,	324
14	32	14 ,,	448
15	25	11 ,,	275
16	20	14 ,,	280
17	40	5 ,,	200
18	50	2 ,,	100
19	45	3 ,,	135
20	40	4 ,,	160
	677	233	6575

FIRST METHOD.

Calculating average width.	Calculating average value.
The sum of the widths sampled, divided by the total number of samples taken, represents the average width.	The sum of the assay values, divided by the number of samples taken, represents the average value.

Second Method.

Calculating average width.	Calculating average value.
Arbitrary stoping width of 36 inches is assumed.	The sum of the total assay inches, obtained by multiplying the width sampled in each instance by the respective assay values, divided by the sum of the number of samples at an arbitrary width, which latter it is intended shall be the average stoping width in any given section of the mine.

Adverting to the schedule foregoing we may now set out the results obtained by these two methods:—

First Method.		Second Method.	
Average width, inches.	Average value, dwts.	Average width, inches.	Average value, dwts.
33·85	11·65	36	18·2

Manifestly there is some fault with methods which give such widely different results, and it may be well to examine wherein the errors lie.

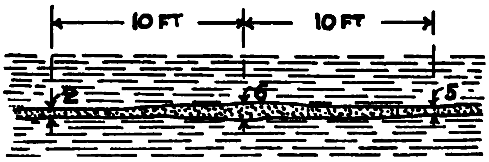


Fig. 42.

For the first method let three samples be taken from a reef as above (Fig. 42), and having the values shown below:—

Sample No.	Width, inches.	Value, dwts.
1	2	150
2	6	40
3	5	10
	13 ÷ 3 = 4·3	200 ÷ 3 = 66·6

The error here lies in assuming that all three samples are the width of the minimum sample. This statement can be proved in the following manner. Let two tons of rock be mined from each point sampled, and assume the total yield of gold to be extracted, then we have :—

2	tons at	150	dwts. per ton, from which we extract	300	dwts.
2	„	40	„ „ „ „	80	„
2	„	10	„ „ „ „	20	„
6				400	

From the total 6 tons we have, therefore, extracted 400 dwts., which is equivalent to 66·6 dwts. per ton, or a similar value to that shown above. If the maximum width be taken as a standard, and all other samples be brought to that standard, then, the simple division of the column, as shown, will give a true average. This can be proved in the following manner :—

Assume, as before, that we mine and treat the following tonnage, and that we extract all the gold contained.

The standard weight being 6 tons, we must include in sections 1 and 3 a portion of barren quartzite to make up the quantity required.

Then, from section 1 we obtain	2 tons	ore	at 150 dwts.	= 300
and	4 „	quartzite.		
„ 2 we obtain	6 „	ore	at 40 „	= 240
3 „	5 „	„	at 10 „	= 50
and	1 ton	quartzite.		
Totals	18			590

From 18 tons of ore mined we have, therefore, extracted 590 dwts., equivalent to 32·8 dwts. per ton. The average arrived at, therefore, by bringing the width of the 3 samples under notice over a standard of 6 inches should be the same as that shown, which can now be proved :—

Sample No.	Width.	Value.	Value on six inches.
1	2	150	50
2	6	40	40
3	5	10	8·3
			98·3 ÷ 3 = 32·8

To prove the error in the second method stated, we may take the following figures from four samples (Fig. 43) :—

Sample No.	Width, inches.	Value, dwts.	Assay inches.
1	16	10	160
2	60	30	1800
3	50	25	1250
4	20	15	300
			3510

Total assay inches, 3510, divided by the arbitrary stoping width of 36 inches = 97·5 dwts. Why is this method wrong? Because in each of the above cases where the reef is above 36 inches, we have first used a factor above 36, for obtaining the total assay inches represented, and have then divided by 36 to obtain an average. Take one instance from the above samples.

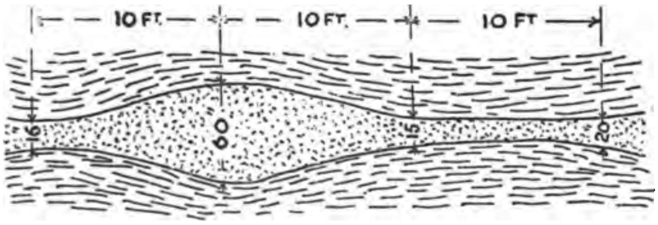


Fig. 43.

Section No. 2 is sampled over a width of 60 inches and assays 30 dwts. The product in assay inches is, therefore, 1800. Now we use the arbitrary divisor of 36, to fix the value of the section, and find that the value is now $\frac{1800}{36} = 50$ dwts., which is absurd. The practical effect of using a divisor less than the factor of width—which with its accompanying value yields a product of assay inches—serves to concentrate the value of a greater width within a lesser, which is absurd, for if it were not, then a deposit 100 feet wide, worth 6 dwts. per ton, which was calculated in this manner, would have a value of 33·3 dwts. on 36 inches—a proposition no one will be willing to support.

The following important deductions from the foregoing statements and illustrations may now be made:—

(1) The assay inch is the product of 1 inch in thickness, multiplied by the assay value over the whole thickness sampled.

(2) In any given portion of a reef, the total number of assay inches is represented by the product of the number of inches sampled, multiplied by the assay value over that thickness.

(3) The total number of assay inches at any given point of a reef, divided by the thickness sampled, gives the average value over that stated thickness.

(4) The total number of assay inches at any given point of a reef, divided by a thickness greater than that sampled, will give the average value over the greater thickness.

(5) The total number of assay inches at any given point of a reef, divided by a thickness less than that sampled, gives a misleading result.

(6) Samples taken at regular intervals, and over an invariable width, are correctly averaged over the stated width, by dividing the sum of the assay result by the number of intervals represented. It is necessary, however, in all cases to standardize the result obtained as over the proposed working width of stope.

In a general valuation of fully developed ore in a mine, the system adopted is based, firstly, upon the assay values shown by

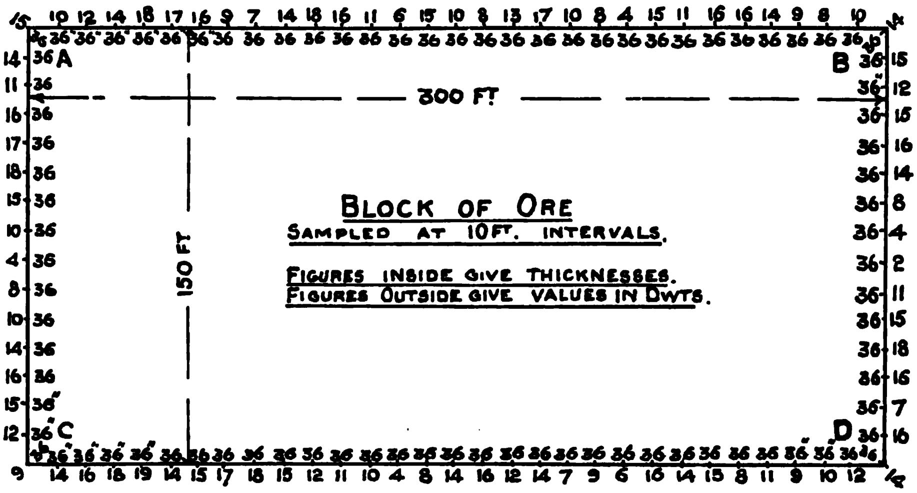


Fig. 44.

the developed blocks, and, secondly, upon the tonnages which the various blocks contain, in the following manner.

Let A B C D (Fig. 44) represent a developed block of ore, sampled at 10 feet intervals:—

The average of the upper level is 12·3 dwts. on 36 inches.
" " lower " 12·4 " " "
" " winze connection at the right is 12·3 on 36 inches.
" " " " on the left 12·7 " "
" value of the levels is, say 12·3 dwts. on 36 inches.
" " winzes " 12·5 " " "

Then 300 feet of level at 12·3 dwts. = 3690 assay feet,
and 150 „ winzes „ 12·5 „ = 1875 „ „
450 5565

$$\frac{5565}{450} = 12{\cdot}3$$

Result—Average value of block 12·3 dwts. on 36 inches. Tonnage contained in block (150 × 300 × 3) ÷ 12 = 11,250 tons.

In another section of the mine the reef may be wide enough to average the values on a width of 48 inches, which would increase the tonnage by 25 per cent. When all the blocks were averaged out as above, they would be brought into column, and calculated out to average as shown below (Fig. 45):—

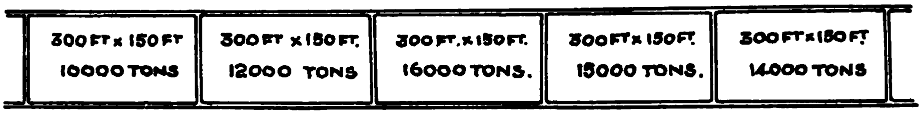


Fig. 45.

10,000 tons having an assay value of 15 dwts. = 150,000 assay tons.
12,000 „ „ „ „ 14 „ = 168,000 „ „
16,000 „ „ „ „ 13 „ = 208,000 „ „
15,000 „ „ „ „ 12 „ = 180,000 „ „
14,000 „ „ „ „ 11 „ = 154,000 „ „
67,000 860,000

$$\frac{860{,}000}{67{,}000} = 12{\cdot}8 \text{ dwts.}$$

Result—Average value of 67,000 tons developed = 12·8 dwts.

Relation of Assay Value as shown on Assay Plans to the Recovery Value.

The ultimate value of "assaying" in a mine is regarded in many instances by the so-called "practical man" as *nil*, for the reason, largely, that there are always wide differences between the average assay result which an assay plan shows, and the value which is finally recovered by the treatment processes; and he has not, as a rule, sufficiently studied the ratio as between the original assay value of ore standing in the mine and the gold recovered, to discover that there is certain invariable relation of the one to the other, provided always that the sampling and averaging have been intelligently performed.

It is popularly supposed, for instance, that a mine which assays on the average 6 dwts. of gold per ton will return that value, sterling, when the ore is treated; and therefore it is stated with some frequency that ores worth 6 dwts. per ton—

equivalent to say 24*s*. sterling—will pay to mine and treat, in places where the working costs are not above 20*s*. per ton. The error of this view lies in the fact that no allowances are made for underestimation of the width of stope which will be carried, and none for the losses which occur in the ore treatment processes. If, for instance, the value of 6 dwts. per ton were correctly arrived at on an estimated stoping width of 36 inches, and if, in actual practice, owing to the occurrence of soft seams, or for any other reason, the narrowest stope possible were 48 inches instead of 36, then at once the value is reduced in proportion 48 : 36 : : 6, or to 4·5 dwts. per ton on 48 inches. To compensate for this possible error, the writer takes 80 per cent. of the value of ore standing in the mine to represent its value when mined and delivered on surface.

The original value of an ore assaying 6 dwts. in the mine is, therefore, by this method, reduced to 4·8 dwts. when it is delivered in the mill bins, provided no sorting is done. The ore is now subjected to treatment, and again practice has proved that 80 per cent. of the value of the ore in the mill bins is about the average amount recovered. Applying that factor to our ore worth 4·5 dwts. in the mill bins, we arrive at a result of 3·8 dwts., which represents the actual extraction we have secured from an original value of 6 dwts. in the mine. Thus, of the total value the amount secured is only 63 per cent., which for safety's sake the writer reduces to 60 per cent. We may, therefore, lay down the invariable rule, that upon reefs, or stoping widths, of 4 feet and under, the estimated recovery value may only be taken as 60 per cent. of the assay value of the ore standing in the mine.

This percentage may, however, be increased by sorting, as elsewhere shown, and by the introduction of a process for the treatment of slimes. By the first expedient it is possible to get an extraction per ton milled of the full value of the ore standing in the mine, but the tonnage milled will only represent probably 60 per cent. of the ore mined and delivered on surface.

By the second method it is possible to increase the amount recovered from the original assay by five per cent.

Grades of Ore required to make Working Profitable.

The "grade" of ore which a mine can be made to produce is dependent largely upon the features which are presented by the actual gold occurrence. In wide reefs, which contain the gold evenly disseminated through the mass, no means of stoping or sorting can possibly be introduced which will have the effect of concentrating the gold value over a lesser width of reef, for the reason that each section is precisely similar. In this class of reef, the gold value as shown by the sampling on a proposed stoping width, must therefore be regarded as the ultimate value of the ore which the mine can be made to yield. If, on the other hand, we are dealing with a very narrow reef, resting upon an easily-mined footwall, the ore proper could be either mined separately by first taking out the footwall rock, and subsequently blasting down the reef as a clean product, or, as an alternative, the ore and a minimum width of quartzite could be stoped in one operation—which is the usual plan—and the resulting product subsequently submitted to a selecting or sorting operation, which aims at the rejection of a certain proportion of the barren quartzite, and an enrichment of the balance which it is intended to mill.

The more general condition in the Main Reef series is, however, neither the very thick homogeneous nor the narrow reef, but rather we have alternating bands of auriferous matter and quartzite, constituting the width which is stoped. In this case the value of the ore as mined is subjected to the modification introduced by sorting. The diagrams here given (Figs. 46, 47, 48) illustrate the possibilities in the three cases :—

In the first instance, A, it is shown that because of the homogeneity of the ore, the values are similar on any thickness of stope up to 5 feet. No system of stoping could, therefore, affect this value, nor could any sorting be done.

In the second instance, B, the values vary as follows:

On 12 inches stoping width, assay value 40·0 dwts.
„ 24 „ „ „ „ „ 20·0 „
„ 36 „ „ „ „ „ 13·3 „
„ 48 „ „ „ „ „ 10·0 „
„ 60 „ „ „ „ „ 8·0 „

In the third instance, C, the values vary as follows :—

On 12 inches stoping width, assay value 25·0 dwts.
„ 24 „ „ „ „ „ 12·5 „
„ 36 „ „ „ „ „ 11·3 „
„ 48 „ „ „ „ „ 8.3 „
„ 60 „ „ „ „ „ 7·2 „

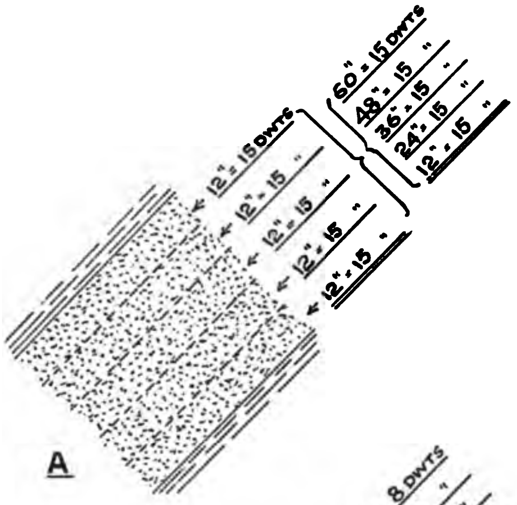


Fig. 46.

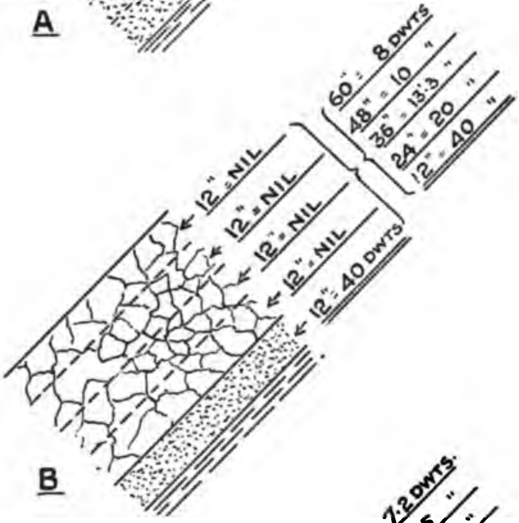


Fig. 47.

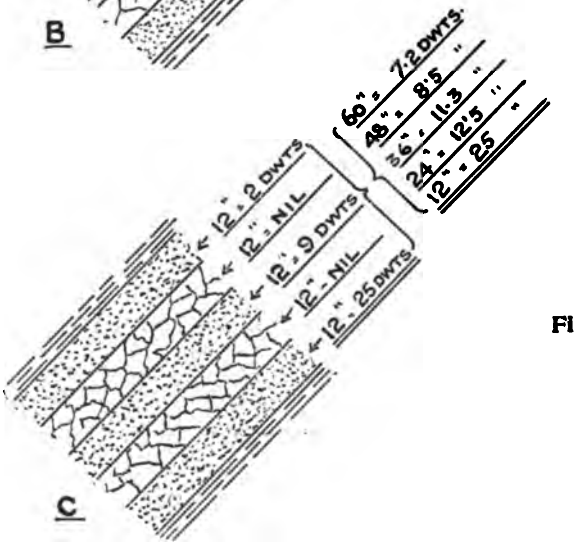


Fig. 48.

It is clear that in cases B and C there is the need for the exercise of an experienced judgment, in determining what width of stope will secure to the company the maximum profit per claim. This immediately raises the question as to the net money value which can be recovered from a mined product assaying 8 dwts. per ton on 60 inches, of which 20 per cent., or 12 inches, contains the whole of the gold. Were sampling methods perfect, and were stopes carried perfectly uniformly, or at least made to average the exact sampled width, or the width taken as the basis upon which the value is arrived at, and were it possible by our reduction and treatment methods to extract 100 per cent. of the gold content, then an average assay of 8 dwts. on 60 inches, not subjecting the ore to a sorting process, would net 33·6 shillings per ton. But experience has shown that sampling gives a result too high by 20 per cent. on account of the practical difficulties of blasting hard rock to the defined lines laid down, and therefore the mined product, instead of being 8 dwts., equal to 33·6 shillings in value—as called for by the sampling—is reduced to 6·4 dwts., equal to 26·8 shillings.

The ore is now at this latter value placed in the mill bins, and experience has shown that of the average value of the ore placed in the mill, an average of only 80 per cent. is recovered. The actual recovery obtained from the gold value of 26·8 shillings, therefore, is $\frac{26{\cdot}8 \times 80}{100}$, or 21·4 shillings, or to sum up the operation, we have only recovered from an original assay value of 33·6 shillings the sum of 21·4 shillings equal to 60·7 per cent. Considering that the average working costs on these fields are 25*s*. per ton, the result would be unpayable. This establishes, therefore, as a general proposition, that ore assaying 8 dwts. in the developed block, is unpayable. We have also determined that a factor of 60 per cent. of the assay value, represents —in the conditions which we are assuming—the ultimate value we may expect to recover. It is easy, therefore, to arrive at the lowest assay value which is requisite in order to pay the average working rate of 25*s*. per ton thus $\frac{10 \times 60}{100} = 6$ dwts. fine gold, which at 4·2 shillings per dwt. exactly balances the working costs. On a mine doing no sorting, the value of the ore in the stoping faces must, therefore, average 10 dwts. per ton, to pay the working expenses.

In the stope, however, upon which we are figuring, the total gold-producing rock only bears a proportion of 20 per cent. to the total mined, the balance being barren quartzite. If it were practicable to stope only the 12 inches of gold-bearing reef, then the recovery value would be $\frac{40 \times 60}{100} = 24$ dwts., equivalent to 100·8 shillings per ton recovery value; but, as in practice it is only very rarely that the conditions are favourable for extracting the narrow width of 12 inches—and then it must be done by mining either the footwall or hanging-wall first—and as in the case under immediate discussion, we assume that not less than 60 inches can be mined, it is necessary to introduce an operation that will effect, more or less, the same result which narrower stoping would achieve. In subjecting the mined ore to sorting, the fact of rejecting a certain percentage of waste rock, and thus concentrating the total value of a given number of tons into a lesser number, provides the equivalent of narrower stoping, which we desire, although, seeing that sorting has a practicable limit, governed on the one hand by the quantity of "fines" which the ore produces, and on the other by the degree of absolute separation of the reef fragments from the accompanying quartzite, by its means it is hardly possible, even in the best conditions, to exceed the equivalent of 40 per cent., which would reduce the 60 inches stope we are considering to 36 inches.

We may now consider the original value of 8 dwts. on 60 inches under this aspect.

Let 100 tons of ore be delivered from the mine into the sorting house, having a value of 6·4 dwts. per ton. Thirty tons of this would pass through the grizzlies as fines, not capable of being sorted, leaving 70 tons to be handled on the sorting table. Of this latter assume that the maximum of 40 tons is rejected as worthless: then the proposition would figure as under:—

100 tons delivered from the mine assaying	6·4 dwts. per ton.
30 „ pass through grizzlies and go to the mill at . .	6·4 „ „
70 tons pass over sorting table, assaying 6·4 dwts., and containing a total of 70 × 6·4	448·0 dwts.
40 „ are rejected as waste, assaying say 1·7 dwts., containing a total of 40 × 1·7	68·0 „
30 tons of sorted ore remain, containing say	380·0 dwts. in all.

Then, of the original 100 tons mined we have—

30 tons fines placed in mill bins at 6·4 dwts.	192 dwts.
30 „ of sorted ore containing	380 „
60 „ of mixed fines and sorted ore contain	572 „

Average value of 60 tons in mill bins $\frac{572}{60}$ = 9·5 dwts., 80 per cent. extraction obtained in reduction and treatment works, 7·6 dwts. Actual recovery value from 8 dwts. mine assay and 40 per cent. sorting = 31·9 shillings.

The costs incurred to obtain this result are, however, considerably higher than the average of 25*s.* per ton, which represents the cost when no sorting is done, and would be somewhat as under, taking some actual figures as a basis—

OPERATING COSTS.

No sorting.			Sorting 40 per cent.		
	Cost per ton milled.			Cost per ton milled.	
	s.	*d.*		*s.*	*d.*
Mining	12	0	Mining	20	0
Milling	6	0	Milling	3	0
Maintenance	2	0	Maintenance	2	0
General charges . . .	1	0	General charges . . .	1	0
Mine development redemption	4	0	Mine development redemption	6	0
Cyaniding	3	0	Cyaniding	3	0
Sorting	0	0	Sorting	1	0
Total . .	25	0	Total . .	36	0

SUMMARY.

Working costs and value of ultimate gold recovered from an average assay of 8 dwts. on 60 inches in the mine.

No Sorting.			Sorting 40 per cent.		
Working cost per ton.	Ultimate recovery per ton.	Percentage of assay recovered.	Working cost per ton.	Ultimate recovery per ton.	Percentage of assay recovered.
25*s.*	21*s.*	60	36*s.*	31·9*s.*	95

The results obtained show that sorting fails to make an ore assaying 8 dwts. in the mine profitable to treat, when costs average 25*s.* per ton, when *not* sorting. If costs were 20*s.* per ton then the results obtained would be as under—

No Sorting.	Cost per ton milled. s. d.	Sorting 40 per cent.	Cost per ton milled. s. d.
Mining	7 0	Mining	11 0
Milling	3 0	Milling	3 0
Maintenance	2 0	Maintenance	2 0
General charges	1 0	General charges	1 0
Mine development redemption	4 0	Mine development redemption	6 0
Cyaniding	3 0	Cyaniding	3 0
Sorting	0 0	Sorting	1 0
Total	20 0	Total	27 0

Summary.

Working costs and value of ultimate gold recovered from an average assay of 8 dwts. on 60 inches in the mine.

No Sorting.			Sorting 40 per cent.		
Working cost per ton.	Ultimate recovery per ton.	Percentage of assay recovered.	Working cost per ton	Ultimate recovery per ton.	Percentage of assay recovered.
20*s.*	21*s.*	60	27*s.*	31·9*s.*	95

In actual practice the increase of cost when sorting 40 per cent. would not be as great as the theoretical advance shown, and therefore the case for sorting is stated at its worst.

Thus far, however, we have only dealt with ore assaying 8 dwts. in the mine, and that value, it has been shown, is unpayable in the conditions that we assume, and would be very doubtful as a profit-earning proposition, even with abnormally low working costs. We have, therefore, now to consider, in case B, whether a stope 48 inches wide, assaying 10 dwts., and costs at 25*s.* per ton milled, is a payable proposition. It is unnecessary to again go in detail through the figures by which we arrived at the working costs, and the percentage of original mine assay recovered, and we may at once tabulate the case, as in the preceding instances.

Assay Value of Ore, 10 dwts. over 48 inches width. Working Cost, 25*s.* per ton.

No Sorting.			Sorting 40 per cent.		
Working cost per ton.	**Ultimate recovery per ton.**	**Percentage of assay recovered.**	**Working cost per ton.**	**Ultimate recovery per ton.**	**Percentage of assay recovered.**
25*s.*	25·2*s.*	60	36*s.*	39·9*s.*	95

Profit per ton milled, no sorting . . . . . . 0·2 shillings
„ „ „ 40 per cent. sorting . . . 3·9 „

If costs were 20*s.* per ton the profit would figure as under—

Profit per ton milled, no sorting . . . . . . 5·2 shillings
„ „ „ 40 per cent sorting . . . 12·9 „

The next instance, in case B, is a 36-inch stope, assaying 13·3 dwts., and let the working costs and percentage of sorting remain as in the last case considered.

Average Value of Ore in Mine 13·3 dwts. Working Costs without sorting 25*s.* per ton.

No Sorting.			Sorting 40 per cent.		
Working cost per ton.	**Ultimate recovery per ton.**	**Per centage of assay recovered.**	**Working cost per ton.**	**Ultimate recovery per ton.**	**Per centage of assay recovered.**
25*s.*	33·4*s.*	60	36*s.*	53*s.*	95

Profit per ton milled, no sorting . . . . . . 8·4 shillings
„ „ „ 40 per cent sorting . . . 17·0 „

If the costs were 20*s.* per ton, the profit would figure as under—

Profit per ton milled, no sorting . . . . . . 13·4 shillings
„ „ „ 40 per cent. sorting . . . 26·0 „

We have in the foregoing illustrations established the following important deductions:—

(1) Ore assaying 8 dwts. and under in the mine is an unsafe mining venture on these fields.

(2) Ore assaying 10 dwts. in the mine is payable, if working costs are low.

(3) The value of sorting varies directly as to the thickness, character, and grade of the reef, and inversely as to the working costs per ton.

In considering the values of stopes in the case C, we may at once dismiss the possibility of stoping payable ore above 36 inches in width. Here, however, we have quite different proportions of reef and "waste" to deal with, for whereas in the case B, already considered, the reef, as to quartzite or "waste," bore only a proportion of 20 per cent. in this instance, out of a total stope of 36 inches, the proportions are 67 per cent. reef and 33 per cent. "waste." The possibility of sorting is therefore confined within much narrower limits in this case than in the case B preceding.

Of the total rock reduced to fines by blasting, the probability is that the proportions of reef to waste are approximately maintained; therefore, in the case under notice of a total of 30 per cent. of fines passing through the grizzlies, the proportions remain one part of waste to two parts of reef. Out of 100 tons, therefore, delivered from the mine to the sorting house, there would be 30 tons of fines, 20 tons being reef and 10 waste. The total waste in the original 100 tons is 33, and deducting 10 for fines, the balance of 23 tons, represents all that will pass over the sorting tables. It is not probable that more than 15 tons would be sorted out, which would equal—as we are dealing with 100 tons—15 per cent. of the total ore, as compared to 40 per cent. sorted out in case B.

The proposition before us is, therefore, What is the ultimate recovery value to be obtained from ore assaying 11·3 dwts. over 36 inches width in the developed block, on a 15 per cent. sorting? Proceeding as in the former case, and taking 100 tons as the basis, we have—

100	tons	delivered from mine worth $\frac{11{\cdot}3 \times 80}{100}$	9·0 dwts.
30	,,	fines pass through grizzly worth 9 dwts. per ton	270·0 ,,
70	,,	ore pass to sorting table containing 70 × 9	630·0 ,,
15	,,	waste rock are sorted out 15 × 1½	22·5 ,,
55	,,	of sorted rock contain	607·5 ,,

Then ore to mill consists of—

30	tons	fines containing	270·0 dwts.
55	,,	sorted ore containing	607·5 ,,
—			——
85	,,	of mixed ore contain	877·5 ,,

Average value of ore in mill bins $\frac{877{\cdot}5}{80} = 10{\cdot}2$ dwts.

Recovery value from 10·3 dwts. $= \frac{10{\cdot}3 \times 80}{100} = 8{\cdot}2$ dwts. $=$ 34·4 shillings.

Theoretical recovery from original assay value, 72 per cent.

Then—Recovery per ton, no sorting . . $\frac{11{\cdot}3 \times 60}{100} = 28{\cdot}1$ shillings.

„ „ „ 15 per cent. sorting $= 34{\cdot}4$ „

In the foregoing instances, the percentage extraction as theoretically arrived at, on various sorting bases, needs a correction for practical application, and below is given the rule which the writer applies:—

To the factor of 60 per cent. of original assay value add half the figure representing the percentage of sorting, and the result is the estimated ultimate recovery value.

Illustration: the developed ore of a mine has an average assay value of 15 dwts. on 42 inches, and it is proposed to sort out 30 per cent. of the ore mined.

Then 60 per cent. represents the factor of extraction when not sorting, and $\frac{30}{2} = 15$, the amount to be added to the first factor. $60 + 15 = 75$, which is the factor of extraction in this instance. Result: extraction without sorting $\frac{15 \times 60}{100} = 9$ dwts.; on 30 per cent. sorting $\frac{15 \times 75}{100} = 11{\cdot}25$ dwts.

From this rule the following factors for estimating the recovery from the original mine assay, on a given sorting percentage, are obtained:—

Per cent. sorting	Factor of original mine assay.
10	65
15	67
20	70
25	72
30	75
35	77
40	80

To avoid complicating the issues involved in the foregoing illustrations, the writer has expressly made no allowances, either in cost of mining, or in variation introduced into differences between estimated assay value of ore in mine, and value at surface, when dealing with wider than average reefs. It is evident that the development redemption charge appearing in the working costs of a company working a wide reef should be less than a company working a narrow reef, for the reason that the tonnage exposed by a given expenditure on mine development would be increased proportionately as to the greater width. Similarly, the expenditure upon labour, materials, and explosives required to extract a given number of tons from a wide reef will generally be less than that required to extract an equal tonnage from a narrow reef.

These allowances must be made to meet the special instance under consideration; but, assuming that they be not figured upon, the omission will at least appear as an error on the right side in actual work.

CHAPTER XIV.

PROBABLE ECONOMIES TO BE EFFECTED WITHIN THE NEXT FIVE YEARS.

THE main heads of expenditure are, as shown by the twelve months' cost analysis elsewhere given:—

Salaries and white wages	32·6	per cent. of the total.
Native wages	21·0	" " "
Materials and stores (other than explosives)	15·0	" " "
Dynamite and explosives generally	10·5	" " "
Coal	9·0	" " "

In considering the value of the future production of the Rand, the question of the possibility of lowering working costs is a very vital one, and it becomes necessary to examine the expenditure under each of the heavy items included in the preceding list, to determine if any saving in respect of these may be effected under the new *régime*.

Rate of White Wages.

There is a decided tendency, on the part of all responsible employers of labour on these fields, to shirk the issues involved in any proposal to reduce the ruling rates of white wages. The reasons are firstly and largely altruistic, there being a strong desire on the part of the financial houses controlling the mines to have a prosperous and contented working community; and secondly, business expediency, there being a feeling that the proposal to introduce reductions would lead to disorganization of the work, and consequent delay and loss. The high rates ruling are in great part a heritage of the past, when skilled labour was scarce, and the demand greater than the supply, and when the cost of conveying supplies to a small community over a great expanse of practically uninhabited country was enormous.

The increased and rapidly expanding market presented for skilled labourers by the mining industry of these Fields, coupled with the high rates of pay, have attracted men from every country in the world, and means of communication being what they are, we are practically in competition with the labour markets of the world; drawing from all a certain number of men, allured hitherward by the higher wages offered. The feeling has, however, never ceased to exist, that residence in this country was at the best but temporary—the unsettled political conditions being mainly, if not wholly answerable for that sentiment—and, consequently, workmen as a class have not been disposed to remove their families and belongings from their native places. All doubt as to permanence and stability now being removed, there will undoubtedly be a big influx of men and families, as well as an army of artisans, from all parts of the world. The net result will be a surplus of labour over demand, and what is inevitable, a lowering of the existing high wages rate. This, however, would not be desired by the heads of the industry, until living costs had been lowered in proportion.

The conditions of existence press here specially heavily upon the married man. Rent, water rates, schooling, etc., are enormously high, and must absorb almost the total amount earned, even when close economy is observed. It is, however, certain that these conditions will be immensely improved in the near future, and the burdens to married and single men become better proportioned. The actual cost to the single man is only slightly higher on the Rand than in the settled mining districts of America or Australia, but all expenses incurred in amusement or luxury of any sort are considerably heavier.

The following list of wages is an average compiled from the total mines of the Rand, by the State Mining Engineer :—

	£	s.	d.	
Managers	92	4	0	per month.
Asaayers	28	3	6	,,
Amalgamators	22	7	3	,,
Battery managers	40	5	8	,,
Cyanide workers	23	6	10	,,
Concentrate workers	21	16	0	,,
Carpenters	25	16	10	,,
Draughtsmen	29	0	6	,,
Engineer (foreman)	38	18	0	,,
Electricians	26	9	4	,,
Engine drivers	25	2	9	,,
Fitters and Mechanics	26	9	5	,,
Gangers	27	1	0	,,
Greaser	18	15	4	,,
Labourers	18	1	8	,,

	£	s.	d.	
Mine foremen	38	11	10	per month.
Mine clerk and storekeeper	22	10	3	„
Miners and trammers	22	9	10	„
Machine drillers	27	9	5	„
Masons	26	7	2	„
Overseers (general)	23	10	3	„
Pitmen	18	7	9	„
Pumpmen	25	2	9	„
Painters	21	16	0	„
Surveyors	30	6	0	„
Secretaries	33	1	2	„
Smiths	26	13	5	„
Stokers	18	15	3	„
Smelters	27	16	10	„
Sundry workers	20	14	2	„
Vanner workers	19	10	0	„

In the settled mining districts of Australia—Bendigo and Ballarat, for instance—the rates for miners, fitters, and carpenters are as under:—

	£	s.	d.	
Miners	11	0	0	per month.
Fitters	15	0	0	„
Carpenters	12	10	0	„

In the United States the rates of pay are somewhat higher, averaging about 3 dollars a day for miners, etc., and 3½ dollars for mechanics.

Stated generally, it may be said that the Witwatersrand mining companies pay their employees 50 per cent. higher wages than the Victorian, and 30 per cent. higher than the American companies.

It is safe to surmise, in view of the altered conditions that will obtain here under the British rule, and specially considering that we are bidding for labour against all the markets of the world, that there will be a levelling process introduced, which will tend to regulate the wage rates on these fields in some manner proportionate to those obtaining elsewhere. Such a reduction will bring within the profit-earning margin many companies that are now beyond that pale; and though to the individual workman the lowering of rates by natural causes may be regarded as unfortunate, still, in the end it benefits the working class as a whole, because the fact that a reduction would stimulate into activity many mines now dormant means that a greater total of workmen would be employed at a good wage after the reduction than was possible before; it means, in short, the greatest good to the greatest number. It has also to be borne in mind that the purchasing power of a sovereign will be greatly increased, before the time arrives when the natural law of supply and demand will have compassed the reduction here indicated, so that in effect the position of the workman will be in no way altered, in respect to his facilities for saving.

If we assume that in five years after the declaration of peace wages will have dropped 30 per cent., it is probable that events will show the assumption not to have been far wide of the mark. On this basis, and calculating the expenditure on white wages as 30 per cent. of the total operating cost, and estimating the average cost at 25*s.* per ton milled, the reduction would amount to 2·3 per ton milled, or rather more than the equivalent of ½ dwt. fine gold. This would total a net saving per month to a company running 200 stamps, and crushing 28,000 tons monthly, of £3150.

Native Labourers' Wages.

This item of expenditure bears to the total operating cost a proportion of about 24 per cent., or taking 25*s.* per ton as the average rate of working, it is equivalent to 6*s.* per ton milled.

The average rate of pay to Kaffirs in the year preceding the war was about 50*s.* per month of 28 working days, exclusive of food, which the companies provide. The sudden demand which arose for native labourers, in a country which had practically no industries, and therefore no schools for the training of native workmen, gave rise also to a rate of pay about three to four times greater than the average paid to natives in the colony of Natal. The steady expansion of the industry, and the increasing requirements for native labourers, has always more than kept pace with the supply available, so that there has always been, as to the total wanted, a shortage of some thousands of natives. This state of things naturally tended to make competition for this class of labour very keen, with the result that the natives were practically masters of the situation, and could demand their own terms. In addition to the actual shortage of supply, the mines were hampered by the effects of drink supplied by illicit dealers to the natives. This resulted in each company having in its compound from 15 to 20 per cent. of its total force either quite incapable of work owing to intoxication, or unwilling to work while suffering recovery. It can readily be imagined that of the natives who did turn out to work, quite an appreciable percentage were indisposed to make an effort to do more than fill in the time. To discharge natives was out of the question, as they would be greedily snapped up at the adjoining mine, and new drafts would behave on the average just as their predecessors.

Lastly, the want of a proper enforcement of the pass system gave rise to a well-organized method of swindling the mining companies, by obtaining deserters from one mine and selling them at high rates to another company in the neighbourhood. The cost of the purchase of natives became in some instances an alarming amount, amounting in one case in the month to 2*s.* 8*d.* per ton milled. The average cost to purchase natives for service with the companies was from £3 to £4 per head, and as there were no means of compelling the native to remain if he preferred to work elsewhere, and no means of properly identifying deserters, a company would sometimes be in the unenviable position of paying, say, £2000 for 500 Kaffirs, and finding at the end of a month that they had practically all deserted, the company having no redress.

Such are in brief the conditions in respect of native labour under which the mining companies have worked in the past. It now becomes necessary to examine in what direction the future promises improvement.

The first requirement to put the native labour question on a satisfactory basis is a well-organized scheme for the recruiting and distribution of native workmen, together with a pass system which will readily lend itself to the registering of natives in their respective compounds, and to the identification and restitution of deserters. The said scheme of organization, to properly perform its function, must be embodied in a regularly constituted authority, vested with powers for making rules and regulations pertaining to all vital matters in connection with native labourers—such, specially, as distribution and rates of pay—and competent to inflict penalties for their non-observance. It will require, for its satisfactory working, entire unanimity amongst the mines, and due and faithful adherence to its provisions on the part of the mine managers.

Granting these postulates, and being free from the machinations of the promiscuous native labour agents, it will be possible to reduce the average rate of pay to 35*s.* per month, and in addition, being clear of the demoralizing and paralyzing effects of the illicit liquor traffic, there will be 10 per cent. more daily workers from the total number of any given compound.

A reduction of native pay to 35*s.* means a lowering of the former rate by 30 per cent., equivalent to 1·8*s.* per ton, or, in the case of a company running 200 stamps and crushing 28,000 tons per month, a saving of £2520 per month.

The question arises as to whether the native labour supply will keep pace with the demand. Reasoning on analogy from past experience, there can be no doubt that it will. The average requirement for the Witwatersrand is about 4 to 5 natives per ton milled per day, so that, with 10,000 stamps running, the mines will require 200,000 to 250,000 native labourers.

As a means to make the mines in some measure independent of native labour, there are doubtless many operations, now performed by Kaffirs, which will in future be done by machines. Many attempts have been made to perfect stoping drills to replace the hand stopers, but so far without achieving success sufficient to warrant their general adoption. The smaller-sized air drills, which are designed for the purpose, are unable to withstand the strain imposed upon the moving parts by the combined shocks caused by air pressure and the force of the blow. Electric drills have been tried, but so far without satisfactory results. The solenoid principle of obtaining reciprocations, which is successful enough when drilling in soft rocks, is unworkable on the Witwatersrand quartzites. It is probable that design in this respect will be in the direction of separate motors connected to the reciprocating movement of the drill by means of a flexible shaft. In any event, there can be no doubt that a stoping machine is practicable and will ultimately be perfected.

Materials and Stores.

According to the analysis of general expenditure, we have under this heading a contribution of 15 per cent. to the total operating costs of the mine, or an equivalent of 3·75*s.* per ton, assuming the average working cost to be 25*s.* per ton milled. The possibility of reducing this charge depends upon two things —firstly, the cost of the material at the port of shipment; and secondly and chiefly, as far as these fields are concerned, upon the rate of freight and transport. The former possibility we may leave out of the calculation, concerning ourselves only with the latter.

It is notorious that South Africa can boast the unenviable distinction of paying the highest rates on freight and railway carriage of any country possessing the volume of commerce which is hers. The late Mr. L. I. Seymour, before the Industrial Commission in Johannesburg, in 1897, tendered evidence in which were included certain comparisons of freight and railroad charges bearing out the statement just made. The writer takes the liberty of extracting from the Industrial Commission's report the following figures given by Mr. Seymour:—

OCEAN FREIGHTS.

(All per ton of 2240 pounds.)

UNION, ALAN LINE, AND AMERICAN AND AFRICAN STEAMSHIP LINE.
(From New York to South Africa direct.)

Freight rates on machinery.		
	Cape Town and Algoa Bay.	East London and Port Natal.
	s. d.	s. d.
Not exceeding 2 tons in weight	32 6	40 0
" " 2½ " "	37 6	45 0
" " 3 " "	42 6	50 0
" " 3½ " "	47 6	55 0
" " 4 " "	52 6	60 0
" " 4½ " "	57 6	65 0
" " 5 " "	62 6	70 0
" " 6 " "	67 6	75 0
" " 7 " "	72 6	80 0
" " 8 " "	77 6	85 0
" " 9 " "	82 6	90 0
" " 10 " "	87 6	95 0
" " 11 " "	92 6	100 0
" " 12 " "	97 6	105 0

NOTE.—Ten per cent. primage is charged on all the above rates, returnable in the form of rebate. The latter is claimed on the 31st of January and 31st of July of each year, and is payable six months thereafter.

SAILING VESSELS.

	East London.	Algoa Bay.
	s. d.	s. p.
1. Mining machinery and cement (iron drums) up to 3 tons . .	18 9	17 3
2. Boilers, etc., 3 tons to 9 tons	23 0	22 3

All per ton weight or measurement (ship's option). Freight net. No primage.

Steamer Freights.

(All up to 3 tons' weight.)

From London, Liverpool, or Glasgow to—

Destination.	Name of line.	I.		II.		III.		IV.	
		s.	*d.*	*s.*	*d.*	*s.*	*d.*	*s.*	*d.*
East London	Bucknall and Clan . .	51	3	38	9	32	6	30	0
East London	Castle and Union (intermediate) . . .	52	6	40	0	32	6	30	0
East London	Castle and Union (mail) .	55	0	41	3	32	6	30	0
Algoa Bay .	Bucknall and Clan . .	42	6	31	3	25	0	22	6
Algoa Bay .	Castle and Union (intermediate) . . .	45	0	32	6	25	0	22	6
Algoa Bay .	Castle and Union (mail) .	47	6	33	9	25	0	22	6

Class I.	Machinery, boilers, etc.
„ II.	Trucks, piping, etc.
„ III.	Rails, soap, earthen pipes, etc.
„ IV.	Bar-iron and steel, cement.

On packages over 3 tons' weight, 5*s.* 6*d.* on every ton or half ton, up to 5 tons.

From 5 to 12 tons, 5*s.* 6*d.* per ton additional. No charge for heavy lifts or heavy measurement.

Sea Freight.

From New York to London, 12*s.* 6*d.* per ton.

From Chicago to London on heavy boilers, *viâ* Kanawha Despatch (Railway, 1000 miles; sea distance, about 3300 miles), 50 cents per 100 pounds, equals 40*s.* 6*d.* per ton of 2240 pounds (weight or measurement *not* at ship's option).

Sea Freight.

From New York to San Francisco, *viâ* Galveston by sea, thence by rail on South Pacific Road to San Francisco—

	£	*s.*	*d.*
In summer, 1*s.* 5½*d.* per 100 pounds, per ton 2240 pounds . . .	1	12	2
In winter	2	7	0

RAILWAY RATES.

For comparison of Indian and other railway rates per mile, *Engineering*, vol. xlii. p. 214, says—

	Charges per ton per mile in pence.				
Class of goods.	India.	England.	Belgium.	Holland.	Germany.
Grain . . .	0·85	1·54	0·79	0·96	1·13
Cotton . . .	1·53	2·77	1·74	1·61	1·61
Cotton goods . .	1·28	2·64	2·22	1·61	2·06
Sugar . . .	0 85	1·12	1·60	0·96	1·32
Average . .	1·13	2·02	1·59	1·28	1·55

English Railway Rates in 1893.

From *Engineering*, vol. lxv. p. 48, January, 1893.

Name of railway.	Old maxima per ton per mile.	New maxima per ton per mile.			
		First 20.	Next 30.	Next 50.	Remainder.
		d.	*d.*	*d.*	*d.*
Great Eastern	2*d.*	1·80	1·50	1·20	0·70
Great Northern	2½*d.*	1·80	1·50	1·20	0·70
Great Western	2½*d.*	1 80	1·50	1·20	0·70
London and North Western	2½*d.*	1·80	1·50	1·20	0·70
London, Catham and Dover	3*d.*	1·80	1·50	1·20	0·70
North Eastern	2*d.*	2 0	1·75	1·50	1·25
South Eastern	2*d.* to 3*d.*	1·80	1·50	1·20	0·75
Taff Vale	2½*d.*	1·80	1·50	1·20	0·70
Lancashire and Yorkshire	2*d.*	1·80	1·50	1·20	0·70

The cost per ton per mile in Germany has been—

Year.	Cost in pence per ton per mile.
1868	1·063
1878	0·911
1880	0·852
1882	0·815
1884	0·792
1886	0·788
1888	0·753
1890	0·749

Ordinary recent rates for machinery over various roads in America, per ton of 2000.

	Total charge.	Distance (miles).	Rate per ton (per mile).
	s. *d.*		*d.*
Chicago to Atalanta, Georgia	47 7	1500	0·3807
„ Syracuse, New York	18 10½	670	0·3381
„ Denver, Colorado	94 5	1041	1·088
„ New York, New York	24 7½	960	0·3078
„ San Francisco, California	90 3	2416	0·4485

For the following named materials, the rate from Chicago to New York in 1892—which practically governed all railway rates in the Eastern States—was per ton of 2000 pounds in car-load lots, distance 960 miles railage :—

Material rate in pence per ton per mile.

Cotton goods.	Stores.	Coffee.	Starch.	Sugar.	Soap.	Machinery.
0·513	0·257	0·257	0·257	0·257	0·2668	0·308

For American railways the total actual cost of carriage to the railways themselves was as follows :—

Railway.	Pence per ton per mile.			
	1887.	1888.	1889.	1890.
Main line branches	0·3298	0·3103	0·3053	0·2955
United railroads of New Jersey . .	0·6303	0·6007	0·5761	0·5663
Philadelphia and Erie railroad . .	0·2659	0·2511	0·2709	0·2511
Lines east of Pittsburg and Erie . .	0·3595	0·3398	0·3348	0·3201

In 1892 all the American railways combined carried 88,241,050,225 tons one mile, for which they received $799,316,042, or an average rate of 0·898 cents; equals 0·4422*d*. per ton per mile.

Comparing the various rates above cited, we have—

German (latest quoted) . . .	0·749*d*.	per ton per mile,	State railways.
English (average)	1·125*d*.	„ „	private railways.
American (average)	0·5422*d*.	„ „	„ „
Machinery, American . . .	0·5126*d*.	per ton per mile,	1·000 per cent.
„ Cape	2·34*d*.	„ „	4·565 „
„ Orange Free State . .	2·34*d*.	„ „	4·565 „
„ Natal	3·04*d*.	„ „	5·931 „
„ Portuguese . . .	4·07*d*.	„ „	7·940 „
„ Netherlands—Cape . .	7·69*d*.	„ „	15·000 „
„ „ Natal .	5·06*d*.	„ „	9·871 „
„ „ Delagoa .	4·27*d*.	„ „	8·330 „
Actual cost coal, Rand tram . .	3·58*d*.	„ „	6·982 „

The standard rate for conveyance of coal from Pennsylvania to New York, over mountainous roads in specially built trucks for coal which cannot be used on the return journey, is 0·3305*d*. per ton per mile.*

The actual cost of coal on the Witwatersrand, from Springs station to the Geldenhuis and Jumpers Deeps, in bulk, is 3·58*d*.

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* In a paper read before the London Chamber of Commerce, Mr. B. H. Thwaite, C.E., a friend of the writer's, has supplied data for rates ruling at present (January, 1902), which make the comparison still more striking. Thus—

Great Britain	¾*d*.
France	0·65*d*.
Germany	½*d*.
United States	¼*d*.

Mr. Thwaite gave also a detailed comparison between the working expenses of a standard English railway and an American one. The comparison, briefly, was that in American practice the average working expenses per ton per 100 miles equalled 24·74*d*., whereas the cost of the standard British line equalled 61·87*d*. The differences in detail were striking, varying from 200 per cent. to 300 per cent. in favour of the American practice.

per ton per mile, or 10·83 times the rate in Eastern United States of America.

MACHINERY SHIPMENTS TO JOHANNESBURG—ACTUAL COST.

(Cost per ton of 2000 pounds.)

	An average lot of mining machinery.			Percentage of total cost.	Cast-iron mortar bases cored and planed, columns, girders, etc.			Percentage of total cost.
	£	s	d.		£	s.	d.	
F. o. b. London or steamer . .	31	5	0	75·64	4	1	10	29·75
Ocean freight (intermediate class) to Port Elizabeth . . .	2	0	2	4·86	2	0	2	14·60
Landing	0	4	3	0·51	0	4	3	1·55
Wharfage ⅜ per cent. on home cost	0	2	4	0·28	0	0	3	0·09
Clearing and forwarding . .	0	3	9	0·45	0	3	9	1·36
Railage to Viljoen's Drift (third class)	5	10	0	13·32	5	10	0	39·98
Viljoen's Drift transport to Johannesburg, including cartage. .	1	13	4	4·04	1	13	4	12·12
Transvaal duty, equals 1½ per cent. on home cost, plus 20 per cent. .	0	7	6	0·90	0	1	6	0 55
Cost delivery at Johannesburg .	41	6	4	100·00	13	15	1	100·00
Difference between home cost and cost delivered at Johannesburg .	10	1	4		9	13	3	
If carriage were 1·13*d.* per ton per mile, as was charged in India in 1886 on grain, sugar, cotton, and cotton goods, the railage would have been.	3	7	3		3	7	3	
Leaving cost at Johannesburg .	37	10	3		9	19	0	
Or difference between home cost and cost here per ton . .	6	5	3		5	17	2	
Saving per ton over present cost .	3	16	1	9·27	3	16	1	27·66

RELATIVE COST OF MACHINERY F. O. B. ENGLAND, CONTINENT, OR AMERICA, AND ERECTED ON THE MINES:—

	Home cost.			Percentage.	Cost erected.		
	£	s.	d.		£	s.	d.
Single drum hoisting engine . . .	1,550	0	0	48	3,218	11	1
Double " " . . .	1,612	0	0	42¼	3,802	14	8
" " " . . .	2,160	0	0	47¼	4,582	1	0
Cornish pumping plant . . .	1,528	0	0	33	4,625	17	3
35-drill compressor complete . .	4,000	0	0	54¾	7,312	0	11
Eight boilers 125 h.-p. each . . .	2,736	0	0	31¼	8,738	9	4
8½ feet by 135 feet chimney complete .	342	0	0	19	1,799	17	0
	1,3928	0	0	40¾	34,079	11	3

Generally speaking, the principal machinery at any mine will be found to cost, when erected, two and a half times its home cost.

No comments on these figures are necessary, excepting that the remark may be permitted that rates which are so evidently above the high-water mark of those ruling in other countries must sooner or later be considerably reduced.

The following extracts taken from the evidence of Messrs. H. Jennings and F. Raleigh, given before the Industrial Commission, illustrate how greatly the prices of material are increased by the prevailing transport charges, and Mr. Jennings' comparison shows what an enormous saving would be effected were it possible to reduce the cost of materials to that ruling at the Alaska Treadwell Mine.

STATEMENT BY MR. H. JENNINGS.

Comparison between Prices of Stores paid by Alaska Treadwell Mine and the Crown Reef.

Article.	Amount used.	Alaska price.			Crown Reef price.		
		£	s.	d.	£	s.	d.
Dynamite	200,089 lbs.	5,134	11	2	17,354	12	2
Fuse	14,314 coils	474	5	0	268	7	9
Caps	75,182	168	5	0	150	8	0
Timber	14,909 cb. ft.	482	0	0	3,168	2	0
Steel, mining	25,519 lbs.	429	16	5	637	18	0
Oils	6,545 gals.	428	3	5	1,309	0	0
Candles	272 boxes	177	15	0	145	0	0
Mortars	2	96	0	0	304	0	0
Mortar liners	58,058 lbs.	756	12	0	1,209	10	10
Cam shafts	3	59	4	0	52	10	0
Guide blocks	120 pair	22	16	0	856	0	0
Shoes and dies	151,922 lbs.	2,178	17	3	2,278	16	8
Screens.	1,300 sq. ft.	109	6	0	97	10	0
Heads	12	49	16	3	78	0	0
Sulphuric acid	328,000 lbs.	1,000	0	0	4,100	0	0
Salt	455 tons	790	8	6	2,733	0	0
Bar iron	63,503 lbs.	275	9	6	529	3	10
Lead	1,461 lbs.	18	4	6	52	11	6
		12,651	10	0	35,324	10	9

If the same proportion exists for the balance of the stores used for the Alaska Treadwell which are not classified above, and which amount to further £3500 exclusive of coal, these total stores, costing £16,100 in Alaska on the above basis of prices, here amount to about £45,100, thus increasing their costs by £29,000, and the total cost per ton at the Alaska Treadwell mine by 2*s*. 2*d*. on the tonnage milled—263,670 tons.

On the other hand, if the Crown Reef Company had been able to obtain its stores last year at the above prices ruling in Alaska, their supplies—exclusive of coal, which actually cost them £85,100—would only have cost them £30,500, which would be a saving of £54,600, or no less than *5s. 6d.* per ton, on their tonnage of 198,236 tons.

STATEMENT BY MR. F. RALEIGH.

RAND MINES, LIMITED, DETAILS OF IMPORTING, COST OF MACHINERY, PLANT, ETC. *via* EAST LONDON.

Description of goods.	Home cost.	Sea freight.	Colonial charges, duty, agency, landing, etc.	Railway.	Z. A. R. duty.	Insurance and commission.	Total cost.	Total importing charges per cent. on home cost.	Railway charges per cent. on home cost.
	£	£	£	£	£	£	£		
Copper plates . .	413	13	2	53	7	14	502	21·55	12·59
Cast-iron base plates	114	53	8	201	2	4	382	235·09	176·31
Wrought-iron pipe .	285	46	9	214	5	11	570	100·00	75·09
Gate valves . .	79	3	1	15	1	3	102	27·85	17·72
Battery post shoes .	26	10	2	38	½	1	77	192·30	142·31
Boiler breechings .	127	79	12	63	2	4	287	126·77	48·82
Netherton iron . .	268	46	9	227	24	9	583	119·79	84·33
Hex. black nuts .	60	9	5	35	6	2	117	93·33	56·66
Two Comet crushers .	880	108	15	265	16	30	1,314	49·20	30·11
Shoes and dies . .	252	27	5	104	5	9	402	59·13	40 87
Mill engine . .	3,656	325	48	806	66	126	5,027	37·47	22·04
Air pump . . .	223	5	2	16	4	8	258	14·79	7·17
Sheaves . . .	184	33	5	44	3	7	276	49·45	23·37
Cornish pumping rig .	1,227	131	20	402	22	41	1,843	50·28	32·76
Condenser . . .	505	121	9	99	9	17	760	50·49	19·60
Amalgam retorts .	50	5	1	17	1	2	76	50·00	34·00
Anchor bolts . .	109	16	2	61	2	4	194	77·97	55 05
Base plates . .	249	114	15	433	5	9	825	231·32	193·90
Steel stamp heads .	700	42	8	160	12	24	946	35·14	22·71
Two cyanide tanks .	99	19	3	71	2	4	198	97·97	70·70
Steel shaft . . .	21	3	½	10	½	1	36	61·90	42·38
Steel shafting and fittings . . .	277	30	5	111	5	10	438	58·12	40·08
One Corliss engine .	416	40	7	105	8	14	590	41·58	25·00
Angle iron . . .	185	49	9	240	3	7	493	166·48	129·19
Pump . . .	51	4	½	7	1	2	65	29·41	13·73
Mill parts . . .	785	70	15	240	14	29	1,133	44·46	30·57
Cast-iron capitals .	20	7	2	34	½	½	54	22·000	165·00
Winding engine and hoisting plant. .	2,414	151	29	396	43	82	3,115	29·08	16·40
Plunger pump . .	230	23	4	87	4	8	356	54·35	37·83
Rope fly wheel . .	145	67	4	60	3	6	285	95·99	46·69
Bolts and nuts . .	46	5	1	15	1	2	70	50·00	30·43
Two cast-iron pulleys	61	3	½	10	1	2	77	27·87	14·75
Bottom balance bob .	18	1	1	5	½	1	26	44·44	27·78
Rubber pump valves .	46	1	½	2	1	2	52	10.87	4·34
Steel shafting and pulleys . . .	504	43	8	123	9	17	705	39·88	24·40
Stamp mill parts .	864	49	9	155	15	32	1,124	30·09	17·94

Description of goods.	Home cost.	Sea freight.	Colonial charges, duty, agency, landing, etc.	Railway.	Z. A. R. duty.	Insurance and commission.	Total cost.	Total importing charges per cent. on home cost.	Railway charges per cent. on home cost.
	£	£	£	£	£	£	£		
Stamp mill parts .	2,274	154	31	587	41	78	3,165	39·18	25·81
Three tanks . .	266	43	8	207	5	9	538	102·63	77·82
Four 8-inch plunger pumps . . .	230	24	4	90	4	8	360	56·09	39·13
Manilla rope . .	91	5	6	13	8	3	126	38·46	13·18
Cast-steel cams . .	780	31	8	118	14	27	978	25·26	15·00
Cornish pumping rig .	1,057	88	31	258	19	36	1,489	41·06	24·40
Spur wheel . .	195	16	3	56	3	6	279	43·08	28·72
Mortar bodies . .	300	49	7	124	5	12	497	65·33	41·00
Steel-wire rope . .	458	27	8	97	8	16	614	34·06	21·18
Pigs of lead . .	12	2	1	8	32	½	55	358·33	65·00
Steam winch . .	505	32	4	78	9	17	645	27·72	15·45
Whiting hoist plant .	262	12	2	41	5	9	331	26·33	15·27
Cast-steel pulleys .	68	11	2	18	1	2	102	50·00	26·47
Stamp mill parts .	797	87	14	246	14	30	1,188	49·06	30·74
Steel chimneys . .	285	64	12	239	5	10	615	115·79	83·51
Machinery . .	811	87	14	226	14	29	1,181	45·87	27·86
Cam shafts . .	340	26	4	100	6	12	488	43·56	29·41
Stamp stems . .	1,150	104	15	394	21	39	1,723	49·65	34·10
Iron piping and fittings . . .	403	60	7	143	7	14	634	54·34	35·23
Auxiliary shaft . .	190	19	3	51	3	7	273	43·16	26·31
Mortar boxes . .	2,800	376	37	975	50	104	4,342	55·10	34·46
Lathe . . .	89	5	1	12	2	3	112	24·27	12·36
Steel pipe joints . .	2,085	249	36	1,238	38	77	3,723	78·60	59·35
Piping . . .	180	26	4	133	3	7	353	94·44	71·66
"Knowles" tank pump	78	2	1	6	1	3	91	16·66	7·69
Portland cement .	82	83	13	385	180	3	756	819·51	469·50*
Liner plates . .	130	7	1	23	2	5	168	29·23	17·69
Wrought-iron pipe .	412	66	9	314	7	15	823	99·75	76·23
Steel crank shaft .	150	42	2	38	3	5	240	60·00	25·34
Boiler . . .	355	82	7	112	6	12	574	61·69	31·55
24-inch stroke column shaping machinery .	85	7	1	13	2	3	111	30·58	15·19
Forty steel rails. .	80	26	5	130	1	3	245	206·25	162·50
Wrought-iron pipe .	886	189	20	732	16	33	1,876	111·74	82·62
Picking table . .	200	24	4	78	4	7	316	58·00	39·00
Winding engine. .	2,135	215	23	405	38	73	2,879	34·85	18·92
Six steel tanks . .	1,021	162	18	586	18	35	1,840	80·21	22·26
One 10-foot boring and turning mill .	759	94	8	169	14	28	1,072	41·24	22·26
Six cast-iron columns	323	207	11	332	6	17	897	177·74	102·79
Angle iron . . .	243	74	9	335	22	9	692	184·77	137·86
Total of above items .	38,156	4628	671	14,034	941	1351	59,748	6298·45	3891·44

* Special Transvaal duty of 3*s.* per 100 lbs.

DETAILS OF IMPORTING COST OF TIMBER, *viâ* DELAGOA BAY.

Descripton.	Home cost.	Sea freight.	Delagoa Bay charges, 4*s.* per ton measure, 4*s.* per ton weight, 3 per cent. duty.	Railage.	Commission and Transvaal duty.	Total.	Total importing charges per cent on home cost.	Railage charges per cent. on home cost.
	£	£	£	£	£	£	£	£
Rough . . .	1700	3363	801	6150	82	12,096	611·53	361·76
" . . .	44	59	22	112	2	239	442·95	254·54
" . . .	16	28	11	55	1	111	593·75	343·75
Selected . . .	65	100	23	202	3	393	504·62	310·77
Tongued and groved	317	335	72	512	28	1,264	298·74	161·51
Doors . . .	86	27	7	30	8	158	83·72	44·18
Windows and frames	181	59	14	60	16	330	82·32	33·15
	2409	3971	950	7121	140	14,591	505 69	295·59

The cost of freight and railage, as given in Mr. Raleigh's list, is 30 per cent. of the total cost laid down on the mine. Taking it that the cost of materials is 3·75*s.* per ton milled, the transport charges figure out at 1·12*s.* per ton milled. It does not appear to the writer too much to assume a reduction in the all-round cost under this head of 50 per cent. within the next few years, or, say, the equivalent of 6*d.* per ton milled. On this assumption the saving to a company working 200 stamps, and milling 28,000 tons monthly, would be £700 per month.

Dynamite.

This commodity (including under the term explosives in general) bears to the total operating cost a percentage of 10·5, or, on the basis of 25*s.* per ton—taken to represent the average working cost on these fields—the equivalent of 2·6*s.* per ton milled. The rates paid for explosives six months prior to the war were as under :—

Gelatine	105*s.* per case of 50 lbs.
Dynamite (70 per cent.) . . .	85*s.* " "

In considering how a reduction will be effected in connection with explosives we may leave detonators and fuse out of consideration, and deal only with the explosives proper.

The present price of gelatine and dynamite in large quantities, f. o. b. English, American, or Continental ports, is approximately as under:—

Gelatine	40*s*. per case of 50 lbs.
Dynamite	30*s*. „ „

Add to this the following charges:—

Freight, railage, agency, etc.	6*s*. 0*d*. per case.
Transit duty, colonial ports, say	2*s*. 6*d*. „
Transvaal duty, say	20*s*. 0*d*. „

Then—

Cost of gelatine, 40 + 28*s*. 6*d*. = 68*s*. 6*d*. per case.
„ dynamite, 30 + 28*s*. 6*d*. = 58*s*. 6*d*. „

Gelatine is only rarely used in stoping work, and would, therefore, not appear against the charge for mining.

As an average the quantity of dynamite required per ton of ore milled is 1·3 lbs., equal, at the former rate, of 75*s*. per case to 1·95*s*. per ton. At the reduced price of 58*s*. 6*d*. per case, and the same consumption per ton milled, the cost would be 1·5*s*. per ton milled, or a saving of 5·4*d*. per ton.

A further saving would be effected in the cost of gelatine used upon development. At the former price of 105*s*. per case, the average cost of gelatine figured out at an average of about 20*s*. per foot driven. This cost would be reflected in the total charge against a given developed tonnage, and later would appear in the working costs under the head of mine development redemption. The cost of development per foot, when gelatine costs 68*s*. 6*d*., will be reduced to 13*s*., a reduction of 7*s*.

The average cost of development on producing mines is about 70*s*. per foot, and as explosives represent about 28·5 per cent. of this the total cost of development will be reduced from 70*s*. to 63*s*., equivalent to a 10 per cent. reduction. If, therefore, under former conditions a charge of 4*s*. per ton for mine development redemption were necessary, the saving in gelatine would bring this charge to 3·6*s*. per ton, a saving of 4·8*d*. per ton milled.

This charge, however, as for mine development redemption, is a very variable quantity, and the figure of saving shown can only be taken as indicative.

Coal.

This item bears to the total operating costs of the mines on the Fields a proportion of 9 per cent.—equivalent, on an average figure of 25*s*. per ton working costs, to 2·2*s*. per ton milled.

In considering the probable reduction in working costs under this head, we shall ignore the possibilities involved in the advent of better economic conditions in their effect upon the cost of coal cutting and production, and confine ourselves to the items of railage, etc.

Under the system formerly in vogue it was compulsory in all but a few instances to have the coal sacked for rail transport. This operation, together with the loss and wear and tear on sacks, entailed a cost of about 1*s.* 6*d.* per ton of coal. The railage charges were as follows:—

1 mile to 10	3*d.* per mile.
10 miles to 20	2½*d.* „
20 „ 30	2*d.* „
30 „ 50	1½*d.* „

With very few exceptions companies were not permitted to have coal sidings, and were, therefore, compelled to transport the coal from the nearest railway station, at costs ranging from 1*s.* 6*d.* to 3*s.* 6*d.* per ton, depending upon the distance. In addition the companies bore the cost of unloading the coal from trucks to waggons or bins, as the case might be, at a cost of about 1*s.* 6*d.* per ton.

Summarizing these costs, we obtain the following:—

Cost of coal at pit's mouth, average	7*s.* 6*d.* per ton.
„ railage, average distance 30 miles, at average rate of 2½*d.* per mile	6*s.* 3*d.* „
„ bagging coal	1*s.* 6*d.* „
„ of off-loading and transport to bins, average	2*s.* 6*d.* „
Total, representing average cost	17*s.* 9*d.* „

The items in which reduction will be effected are the railage charges, cost of bagging, and cost of transport.

Railage.—The short distances to be hauled, and the fact that the coal trains run empty on the return journey, will always make the coal traffic distinctly expensive; and bearing in mind the conditions of the country after the war, and its need for raising revenue from every legitimate source, it is not probable that the average rate of 2½*d.* per ton per mile in vogue before the war will be reduced by more than one penny per mile, or to a rate of less than 1½*d.* per ton per mile. The net saving on this head will, therefore, be, for the average haul of 30 miles, 2*s.* 6*d.* per ton.

Bagging Coal.—The introduction of bottom discharge trucks of large capacity for carrying bulk coal, and of side-tipping trucks for the smaller sidings, entirely obviates the necessity for bagging. This charge will, therefore, be entirely done away with in future, resulting in a saving of 1*s.* 6*d.* per ton of coal.

Cost of Unloading and Transport.—This item will be reduced in almost every instance, by reason of the fact that the authorities are building or have built, or have in contemplation, sidings to every important mine, the latter then arranging that the trucks shall discharge direct into the bins of the various large steaming stations. The average cost under former conditions has been placed at 2*s.* 6*d.* per ton—a very low estimate. The saving that can be effected will be the difference in the interest on the capital outlay involved in sidings and bins, *plus* the cost of the carriage over the siding. If we assume that the average cost to each company for sidings is £1500, and the cost of bins £2000, the total annual interest charge is £210, or for a company working 100 stamps, and using, say, 15,000 tons of coal per annum, the interest charge is 3·3*d.* per ton of coal.

It may be reckoned that each company will be charged with running over one mile of siding, and assuming a high rate of 6*d.* to cover terminal charges, shunting, etc., the total cost would be 6*d.* per ton per mile. This charge, taken with the interest charge, amounts to 9·3*d.* per ton per mile, as against 2*s.* 6*d.* formerly—a saving of, say, 1*s.* 8*d.* per ton of coal.

Summarized, the estimated saving is as under :—

Railage	2*s.* 6*d.*	per ton.
Bagging coal	1*s.* 6*d.*	„
Unloading and transport	1*s.* 8*d.*	„
Total	5*s.* 8*d.*	„

To a company milling with 100 stamps and burning 1500 tons of coal per month, and crushing 14,000 tons of ore per month, the net saving is £425, or 7·3*d.* per ton milled.

Total Estimated Saving in Working Costs.

This (it will have been seen) works out as follows :—

White wages	2·3*s.*	per ton.
Native wages	1·8*s.*	„
Materials and Stores	0·5*s.*	„
Dynamite	0·45*s.*	„
Coal	0·61*s.*	„
Total saving	5·66*s.*	„

As a set-off against this saving, we have the increased working costs in deep-level mines, elsewhere considered.

Economical Motive-power.

Prior to the discovery of the possibilities of using blast-furnace gas in internal combustion engines, the power-capacity of this type of motor rarely exceeded 150 H.-P.; but engines are now in use developing 1000 H.-P., and the enormous economies obtained are producing quite a revolution of opinion in favour of this type of motor.

The advantages of gas power are numerous and important: in regular working, an economy of from 200 to 300 per cent. compared with steam practice is easily obtainable; and the facts that the character of the water used is one of indifference, and that the supply of water for a given power-capacity is extremely small compared with that required for steam-engines, are alone sufficient to establish the economic superiority of the system. Again, the maximum pressure established in the plant producing and dealing with the power gas never exceeds a few inches of water, so that the possibility of a dangerous explosion is almost remote. In addition, the labour associated with the supervision of the working of the plant is almost insignificant, and the removal of the clinker can be effected by Kaffir labour.

The depreciation of the treatment plant, with reasonable care, could be satisfied by 1 per cent., as compared with the 10 per cent. which should be charged against high-pressure steam-boiler plant.

In the most modern gas-engines and gas-plant, designed by Mr. B. H. Thwaite, C.E., the gas-engines can be applied for driving electric machinery. With regard to the character of the fuel, almost any character is now available. With respect to first cost of the plant, a comparison with high-class steam condensing engines and high-pressure boilers shows that the cost (especially in a mining country) would be in favour of the gas power. There is no doubt that in the future we shall see the almost exclusive adoption of power gas plants, established at convenient points in the coal fields, from which the energy will be transmitted under high pressure to the different mines.

POSTSCRIPT *to page* 163.—TOTAL ESTIMATED SAVING IN WORKING COSTS. Since the table summary under the above heading was written, an Association for the Recruiting and Distribution of Native Labour has been formed, and its provisions are such, that to some Companies, especially those situated within the town radius, there will be considerably less saving than that indicated in the item—

Native wages	1·8*s.* per ton.

Under the new conditions, Companies must pay a capitation fee of £2 to the Native Labour Association for every native registered in their books, whether they seek work voluntarily or are distributed by the Association, and have also to pay the monthly pass-fee of 2*s.* per head. Formerly, Companies centrally situated generally obtained all the natives they required without purchase, and the charge for the monthly pass was deducted from their earnings.

To some Companies, therefore, the reduction of native wages will not mean more than from ninepence to one shilling per ton milled, and the figure of 1·8 shillings as shown in the table summary must be taken to represent the most favourable case, whilst an average of 1·0 to 1·3 shillings will probably represent the general saving.

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